Trow

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CONTRACTOR:
TROW LIMITED

STUDY TO DETERMINE SURFACE CROWN PILLAR MECHANICAL AND STRUCTURAL PROPERTIES
DSS FILE NO. 003SQ-23440-8-9063
PHASE III - FINAL REPORT - DATA EXAMINATION

PREPARED FOR:

DEPARTMENT OF ENERGY, MINES & RESOURCES
CANMET - MRL DIVISION

85 + 15 figs

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Project: S02095R
October 10, 1989

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TABLE OF CONTENTS

1.0 INTRODUCTION 1

2.0 STRUCTURAL DISCONTINUITIES 2

3.0 FAILURE MECHANISMS AND STABILITY EVALUATIONS 4

3.1 Empirical Models 5

3.2 Numerical Models 9

4.0 SUMMARY 10

APPENDICES

Fracture Frequency Distributions Appendix 1

Core Logs Appendix 2

Simple Empirical Models Appendix 3

Mathews Method Appendix 4

Mine Cross Sections Appendix 5

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S02095R

October 10, 1989

Department of Energy,
Mines & Resources
CANMET - MRL Division
555 Booth Street
Ottawa, Ontario
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ATTENTION: Mr. Denis Labrie, M. Sc. A., Eng.

Dear Sirs:

**STUDY TO DETERMINE SURFACE CROWN PILLAR
MECHANICAL AND STRUCTURAL PROPERTIES
DSS FILE NO. 003SQ-23440-8-9063
PHASE III - FINAL REPORT - DATA EXAMINATION**

Subsequent to the Crown's acceptance of the Phase II[1] report and authorization to proceed with Phase III, given on April 5, 1989, Trow Ontario Ltd. proceeded with the work outlined for this phase. This final report summarizes the data as required by Task I, expand upon the statistical work done on structural discontinuities asked for in Task II and a review of modelling and dimensioning techniques for crown pillars as outlined in Task III.

1.0 INTRODUCTION

This interim report covers the following items. The first section outlines the collected data in log form so that structural, strength and deformational characteristics are presented side by side. These have been plotted on individual cross sections as required by your office and presented in Appendix 5.

This section also provides the mapping of all discontinuities on a lower hemisphere equal area Schmidt projection. No correlation was indicated between individual families of joints or concentration of joints and rock type. As such, only poles for each borehole have been plotted

[1] *Piciacchia, L., 1989, "Study to Determine Surface Crown Pillar Mechanical Properties, Phase II report".*

with no distinctions for rock type being made. Contour plots of pole concentrations have been provided in order to help identify individual families of joints at each mine site. These plots have been represented relative to assumed top of borehole as borehole runs were not surveyed and core was not oriented relative to true top of borehole. As such true north is not known.

The fracture spacings, between individual fracture, has been statistically analyzed for each borehole to provide us with a mean and modal joint spacing for each borehole.

Lastly, a discussion as to possible failure modes has been provided and an evaluation of each crown pillar has been made using various analytical and numerical techniques.

2.0 STRUCTURAL DISCONTINUITIES

Individual structural discontinuities, for each mine site, have been mapped on Schmidt equal area nets, Dwgs. 1 to 4. These were subsequently contoured and individual families of joints identified, Dwgs. 5 to 8. Average dip and dip direction values for each family of discontinuities were subsequently used to plot great circles for each family of joints, Dwgs. 1 to 4. Typically two families of joints were identified; Table 1 summarizes our findings in terms of major and minor joint sets, based on joint concentrations.

TABLE 1

Summary of Major and Minor Joint Sets

Mine	# of Major Sets	# of Minor Sets
Pamour	1	1
Bousquet	1	1
Belmoral	2	3
Chimo	2	0

In addition to evaluating individual families of joints in terms of frequency of occurrence, the frequency of individual fractures, independent of orientation were also evaluated for each borehole. This data has been summarized in Table 2, and statistical histograms have been provided in Appendix 1.

TABLE 2

Summary of Statistical Analysis of Fracture Frequency

Mine Site	Mean (ins.)	Standard Deviation (ins.)	Modal Value (ins.)
Pamour			
P1	3.77	3.70	2.5
P2	6.9	5.3	4
P3	3.80	3.50	3
Bousquet			
B1	3.72	3.72	1
B2	4.95	5.17	3
B3	4.39	4.56	1
Belmoral			
BM1	10.18	10.95	3.25
BM2	6.49	5.64	2.5
BM3	9.59	9.39	3.85
Chimo			
C1	5.98	5.57	3.5
C2	5.75	5.63	3.5
C3	5.76	4.82	2

It is noted that consistent mean and modal values were observed at each site and the standard deviations were typically of the same magnitude as the mean. This latter observation implies that for individual boreholes the actual fractures can be much more widely spaced than indicated by the mean. It is also noted that the plotted histograms indicate a skew to the right of the mean in all cases. Thus the use of

these mean values would provide a conservative approach, for design purposes in all four cases. The modal values, as indicated in Table 2, tended to be even more conservative than did the mean values.

In addition our core logs have been revised to include the available field and laboratory test data related to strength and deformational[2] properties. These have been provided in Appendix 2.

3.0 FAILURE MECHANISMS AND STABILITY EVALUATIONS

Failure mechanics, in general, are related to two factors, stress and structural environments. However, in the instance of near surface crown pillars, where stress levels are generally accepted as low, structural failures are generally more predominant. This is not to say that the influence of stress is not important in structurally controlled failures. In-situ stresses provide a key source for interlocking existing blocks by increasing normal stresses and hence shear resistance. This optimistic outlook on the influence of in-situ stress assumes that the jointing patterns are favourably oriented within the stress environment. Stresses in the worst cases can contribute to failure by contributing to the driving forces along the planes of weakness. In either cases the influence of stress in inducing or preventing structural failures is certainly an important consideration.

The most important consideration for determining the likelihood and possible mechanism of failure is the relative spatial and geometric orientation of existing jointing structures and mine opening. For example, if both the mine opening and the predominate jointing set are near vertical then the most likely failure mode would be toppling on the hangingwall and footwall, while the jointing in the back would be susceptible to fail by a plug/drop mechanism. If however, the mine openings were near horizontal in the same jointing environment, then one would only be concerned with the plug/drop mechanism in the back. For relative orientations between these two extremes, failure mechanisms including combinations of plane, toppling and plug/drop mechanisms may occur depending on the specific relative dip

[2] *Gorski, B., Murray, G., Schlueter, S. 1989. "Strength Determinations of Belmoral, Pamour, Chimo and Bousquet Mine Rocks". MRL 89-22 (INT)*

and strike angles between the joints and openings being considered. It is not the scope of this work to expand upon failure mechanisms any further. However, it is noted that the literature shows a surprising lack of information with regards to assessing possible failure mechanisms and assessing limiting cases for various mechanisms, (i.e., toppling failures may occur for relative dip angles between 0° to 10° and actual dip angles between 75° and 90°).

Several empirical and numerical models are available for assessing ground stability. Each of these general groups provides some pros and cons with respect to their usage in assessing the stability of near surface crown pillars.

For instance numerical approaches such as finite element methods often require input properties, such as material modulus and strength characteristics, these are difficult to assess for the in-situ rockmass. The various rockmass classifications systems do not readily lend themselves to input into such models. Conversely, most empirical models are based, fundamentally, on specific rockmass classification systems.

The following paragraphs will first expand upon some of the noted limitations of both numerical and empirical techniques. It will follow with specific assessments of stability using four empirical and one numerical method. It will then look towards other available, applicable techniques for evaluating crown pillar stability.

3.1 Empirical Models

The empirical models discussed in this section are all linked to a particular rockmass classification system. As such these empirical models are prone to the same assumptions and limitations as the classification system they use. Thus, it is not possible to discuss these empirical models without first examining the rockmass classification system these empirical models use as an integral part of the design process.

Seven rockmass classification systems were examined, these ranged from the simple Deere[3] classification to the more complex NGI[4] and CSIR[5] systems. It is noted that these systems were initially intended to provide a representation of rockmass properties. Each system helps describe the rockmass to varying degrees, however, none of the systems reviewed, adequately incorporated the interaction between mining geometry, orientation and rockmass characteristics.

After reviewing these classification systems, three simple trial empirical models were applied to each mine investigated. The three models were the Merritt[6] system, NGI tunneling system and the CSIR standup time. The design charts used are illustrated on Dwg. 9. These three systems incorporate the Deere-RQD, NGI-Q and CSIR-RMR indices, respectively, therefore, each of these systems will be examined in a little more detail.

Deere's classification is a quantitative index of rockmass quality based upon core recovery from diamond drilling. This method is prone to bias arising from drilling techniques and is limited as it only incorporates fracture frequency and does not account for fracture orientation, effect of stress and water, etc. Thus these same limitations apply to the Merritt design system.

[3] *Deere, D.U. (1964). Technical description of rock cores for engineering purposes. Rock Mechanics and Engineering Geology, Vol. 1, No. 1, pp.17-22.*

[4] *Barton, N., Lien, R., and Lunde, J. (1974). Engineering classification of rockmasses for the design of tunnel support. Rock Mechanics, Vol. 6, No. 4, pp. 189-236.*

[5] *Bieniawski, Z.T. (1974). Geomechanics classification of rockmasses and its application in tunnelling. Proc. Third International Congress on Rock Mechanics, ISRM, Denver, Vol. 11A, pp. 27-32.*

[6] *Merritt, A.H. Geologic prediction for underground excavations. Proc. First North American Rapid Excavation and Tunnelling Conference, AIME, New York, 1972, pages 115-132.*

The NGI system, on the other hand, attempts to account for cross jointing, joint roughness and alteration, water and stress conditions. It further incorporates the RQD index, however, joint orientation is not considered in this particular system.

Lastly the CSIR system involves a combination of factors such as RQD, strength of intact material, joint spacing, joint and groundwater conditions. Several modifications to the derived index are then made to account for joint orientation relative to the opening. This system however does not account for the effect of in-situ, induced stresses.

These three classification system are used as an integral part of the three design concepts illustrated on Dwg. 9. It is further noted here that each of these design concepts were geared towards design of tunnels, as such no direct account is taken of the dimensions and stability of the superincumbent rock above the opening (crown pillar). Therefore, at best, only an indirect measure of crown pillar stability can be made with these models.

The results of this phase of the modelling is summarized on Table 3 and detailed calculations and summary of assumptions have been provided in Appendix 3.

It is evident from this position of the modelling that the CSIR standup time yields totally unreliable and unreasonable results. This is principally due to the fact that the definition of active span is based on Lauffer's[7] interpretation which does not account for depth and dip length openings.

It is difficult to comment either on the Merritt or NGI classification systems as very little back analysis data is available. It is noted, however, that a cave-in, to surface did occur at Bousquet and

[7] *Lauffer, H. (1958). Gebirgsklassifizierung fur den stollenbau.* *Geologie und Bauwesen, Vol 24, No. 1, pp. 46-51.*

that the NGI system predicts that no support be implemented. In fact the NGI system indicates no support is required in all cases. Conversely Table 3 indicates that the Merrit system predicts the requirement of some form of support in all cases with the exception of Chimo.

In addition to these tunneling empirical models a modified version of the Mathews[8] method was used to assess the stability. This design system is based on a modified NGI-Q rating and has been developed to be as assistance in stope dimensioning at depth. The system has been modified by applying lower levels of initial stress than normally would be applied. This was done to account for the shallow depths involved. An elasto-plastic finite element program was employed to assess induced stresses.

This method uses a selection of geotechnical factors which are combined to provide a stability number which is the plotted against a shape factor to emprically assess the stability of surfaces which bound open stopes. The stability number used accounts for rockmass quality, the state of stress and the orientation of exposed surfaces, while the shape factor accounts for the shape and size of the opening. This method's biggest limitation is its ability to handle single openings only. This method has been applied to each case study and the summary of our findings are outlined in Table 4 and Dwg. 10. Appendix 4 provides all the design curves used and outlines the calculation procedure applied to each case study. The summary in Table 4 indicates that none of these sites are potentially caving, however, several potentially unstable conditions exist. This was particularly true for Bousquet where the hangingwall, footwall and back were in a potentially unstable state. This would certainly concur with the observed cave-in at this site.

It is also interesting to note that Pamour has had several hangingwall cave-ins and footwall slippages in the past. Furthermore, the hangingwall instability problems predicted for Belmoral are well known and amply documented.

[8] *Mathews, K.E., Hoek, E., Wydlie, D.C. and Stewart, S.B.V., 1981, "Prediction of Stable Excavation Spans for Mining at Depths Below 1000 metres in Hard Rock".*

In summary this method seems to be suitable for use in crown pillar studies, however, some additional work would be required to refine the model for use in near surface locations. In addition, many more back analysis should be carried out in a variety of environments in order to thoroughly assess the limitations and the applicability of this model. Lastly it is also interesting to note that the simple Merritt system provides predictions which concur with the more complicated Matthews method.

3.2 Numerical Models

The applicability of numerical models should be taken in the context of the particular type of models being considered. For instance, a stress analysis model, linear or non-linear would have limited applicability in the assessment of crown pillar behaviour for two particular reasons. Firstly, pillars are often jointed and fractured so that many of the stress analysis models cannot precisely represent such materials, therefore they fail to represent the rockmass adequately. Furthermore, and more importantly, the stress levels near surface are typically low thus stress levels near surface are typically low thus stress induced failures are uncommon, unless overburden depths are excessive or particularly high horizontal stresses exist. This has been illustrated using a simple elasto plastic finite element model, Dwg. 11. This drawing is an illustration of the situation at Chimo, it indicates that stress levels in the crown pillar are typically maintained below 2 MPa. Under these circumstances stress analysis models will always predict safe situations as they do not account for jointing and the movement of such joints into the existing opening. This not withstanding stress analysis models can be used to provide a first estimate of induced stress to be used in conjunction with empirical methods such as the Matthews method.

The second group of numerical codes consists of discrete block models. These models are geared towards jointed rockmasses. As such, the rockmass can be modelled as discrete blocks which are allowed to move relative to each other and the opening. These models are therefore

suitable for representing relative and actual joint orientation. Trajectories of movements from such modes can therefore be used to assess modes of failure. It has been our experience that discrete block models are the most applicable for modelling jointed crown pillars.

4.0 SUMMARY

This report has attempted to summarize laboratory and field data for each case study investigated. It has further investigated the use of numerical and empirical models for assessing crown pillar stability.

The statistical analysis on fracture frequency indicated frequencies with means in the order of 5 inches and modes in the range of 3 inches. It has also been noted that the fracture frequency distribution curves are typically skewed to the right of the mean. Furthermore, the standard deviation indicated a wide scatter fracture frequency this indicating that in some instances the fracture spacing can be wider than indicated by the mean.

The investigation of applicable models indicated the poor applicability of empirical models such as the CSIR and NGI systems. It was noted that the Matthews method held some interesting potential for evaluating crown pillar stability and that the predictions from this method seem consistent with predictions using the Merritt system.

Numerical models such as elasto plastic or simple elastic models should serve only as a means of assessing induced stresses.

In reviewing these various models, it is our opinion that the most applicable technique for assessing stability and failure modes in jointed rock mass is discrete block models.

Should you have any questions, please do not hesitate to contact the undersigned at this office.

Yours very truly,
TROW ONTARIO LTD.

LICENSED PROFESSIONAL ENGINEER
L. PICIACCHIA
PROVINCE OF ONTARIO

Luciano Piciacchia, Ph.D.,P.Eng.
Sr. Rock Mechanics Engineer

LP:lstp
Encl.
Dist: Department of Energy,
Mines & Resources
CANMET - MRL Division (8)

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TABLE 3

Summary of Simple Empirical Methods

Mine	Stope Width (m)	Stope Height (m)	O/B Depth (m)	Merritt System	NGI	CSIR Standup Time
Pamour	9.2	38.13	6.1	pattern bolting on all walls	no support required	2½ months
Bousquet	5	40	15	pattern bolting HW/FW	no support required	1 day
Belmoral	3.05	45.8	6.1	pattern bolting on HW	no support required	10 days
Chimo	4	40	30	no support required	no support required	2½ months

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TABLE 4

Summary of The Modified Matthews Method

Mine	Stable Zone	Potentially Unstable Zone	Potentially Caving Zone
Pamour	back, ends	hangingwall, footwall	---
Bousquet	ends	hangingwall, footwall, back	---
Belmoral	ends, footwall back	hangingwall	---
Chimo	ends, hangingwall footwall	back	---

Throw

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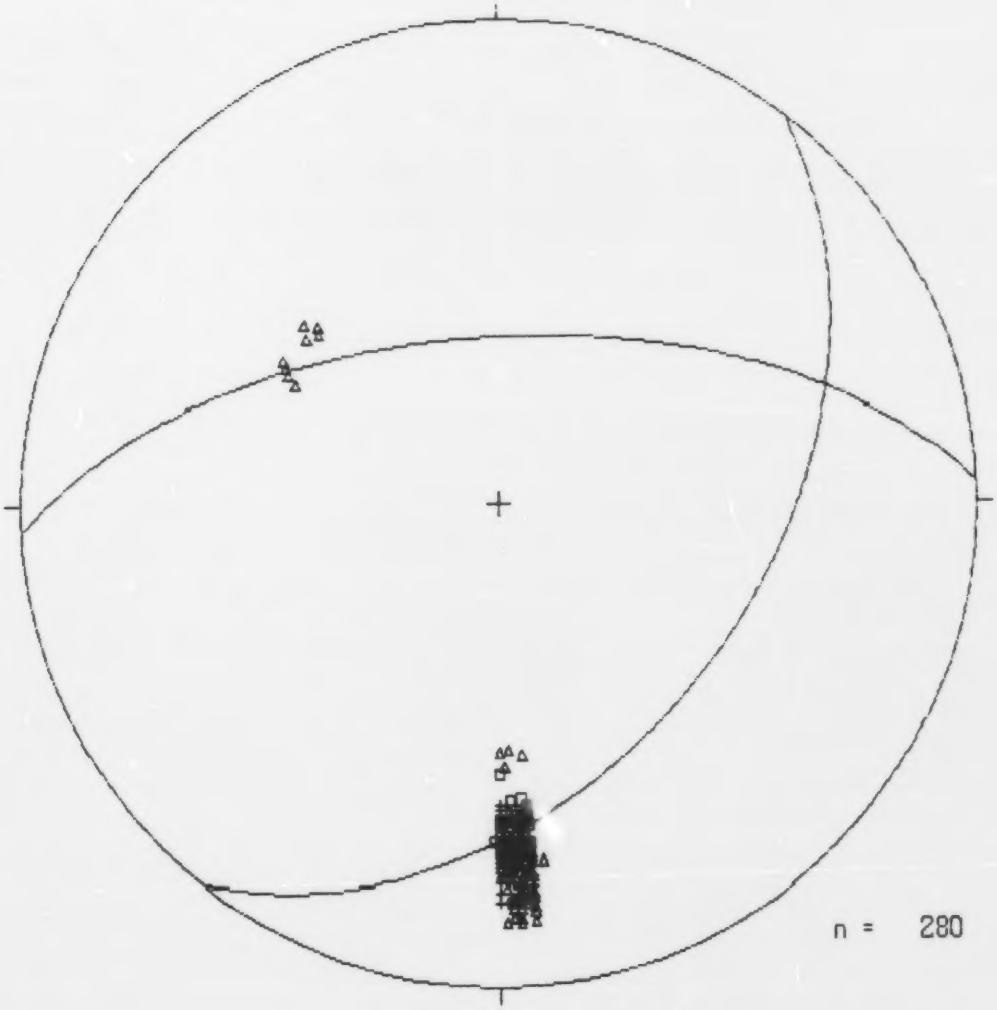
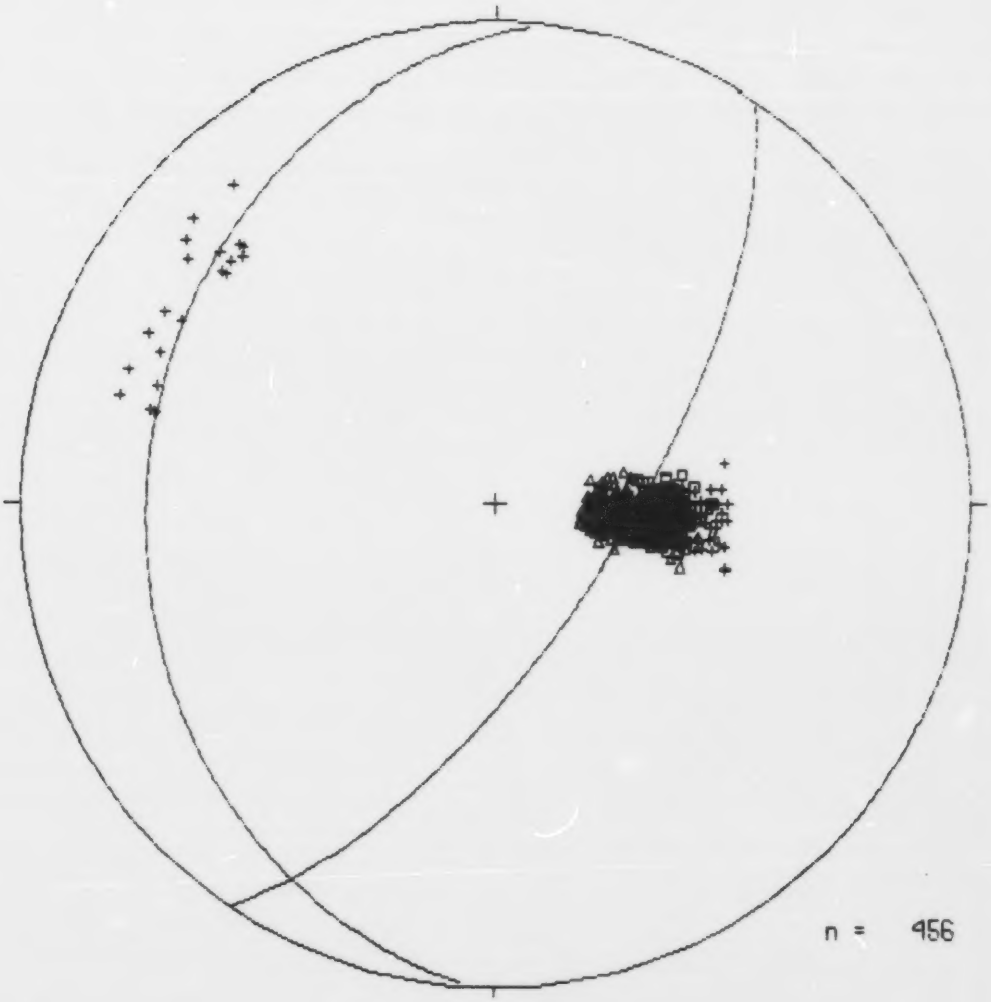
Dwg. No. 1

Trow

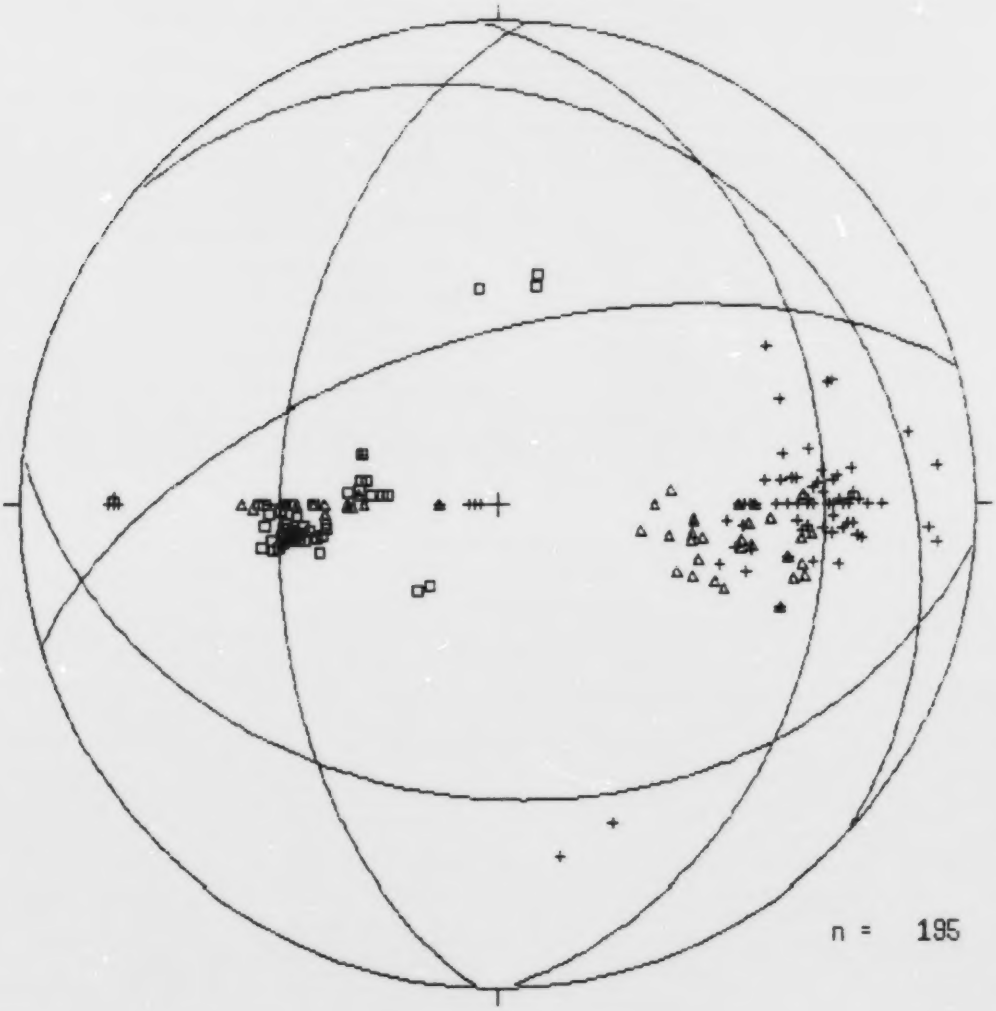
Pamour Mine



Bousquet



Belmoral Mine



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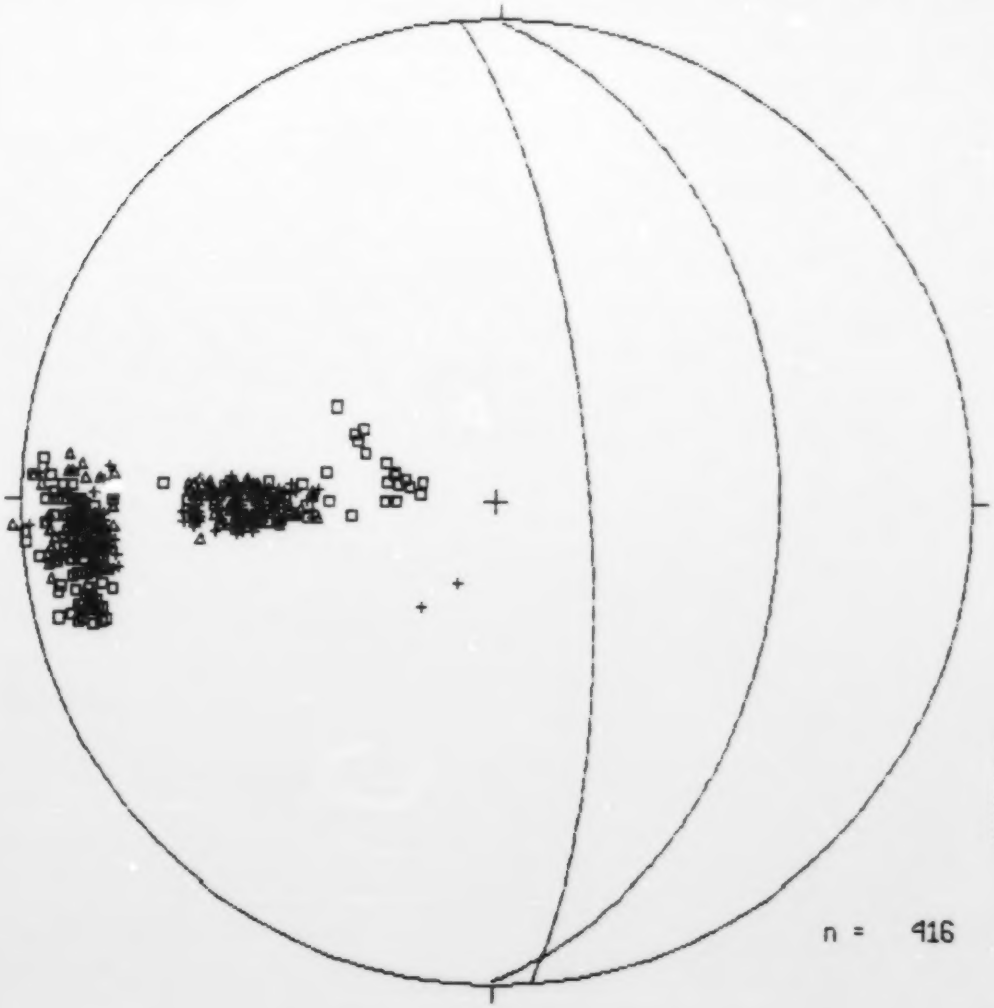
Dwg. No. 4

Trow

Chimo Mine



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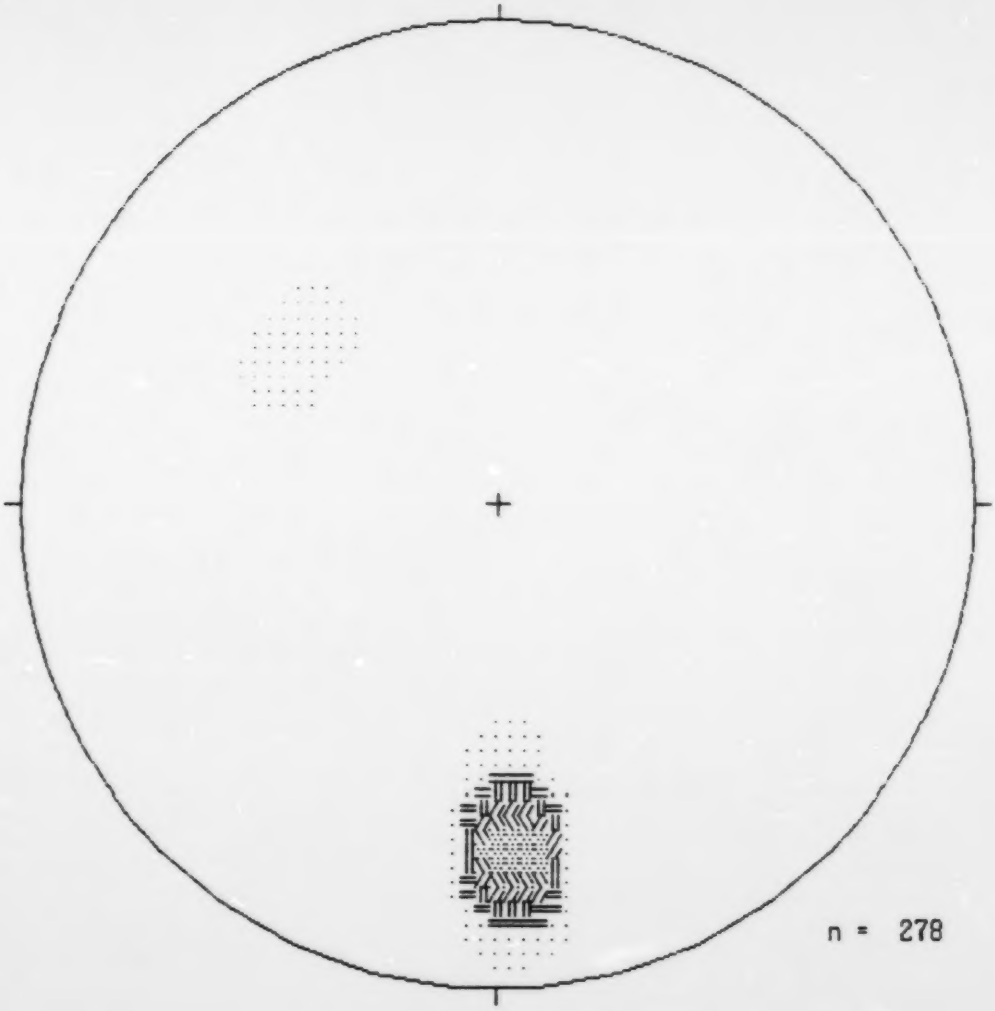
Dwg. No. 5

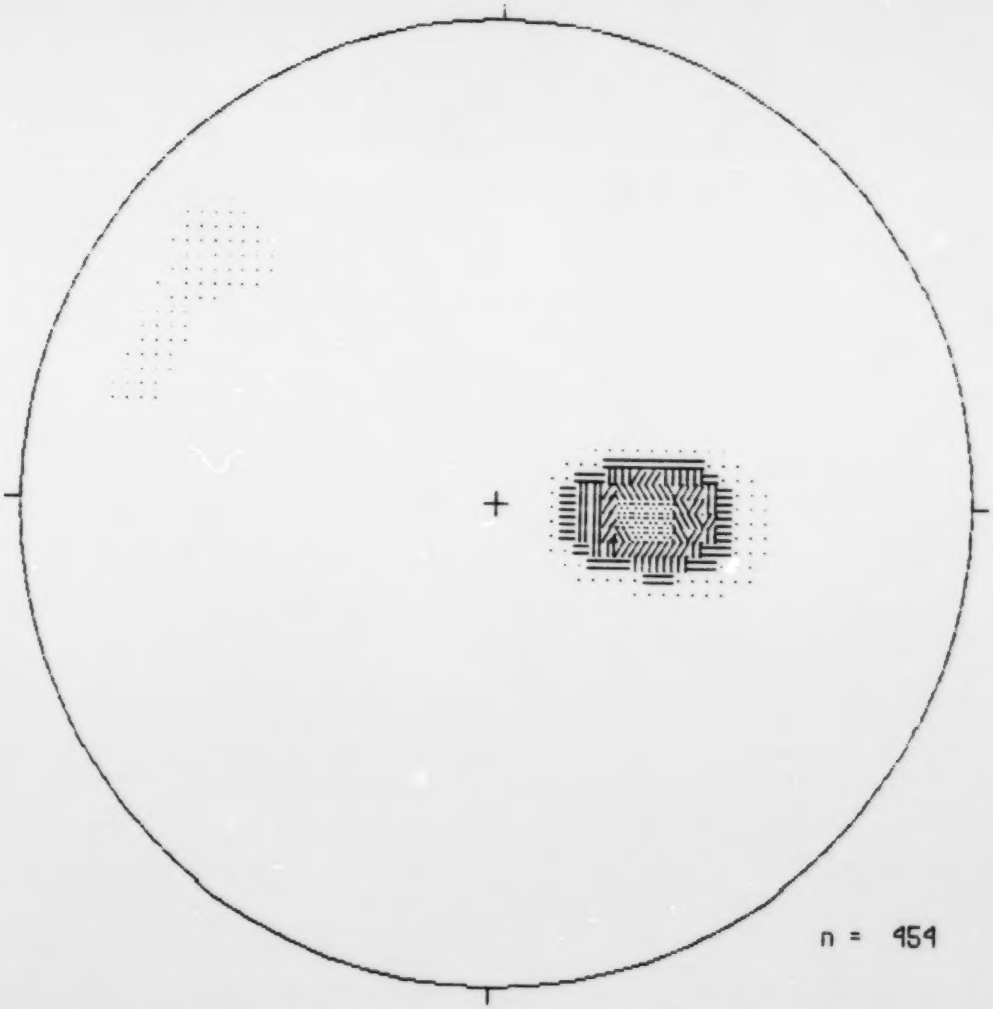
Pamour Mine



0.4-14.4% 14.4-28.4% 28.4-42.4% 42.4-56.4%

56.4-70.4% 70.4-84.5%

Bousquet



0.2-12.2% 12.2-24.2% 24.2-36.2% 36.2-48.2%

48.2-60.2% 60.2-70.9%

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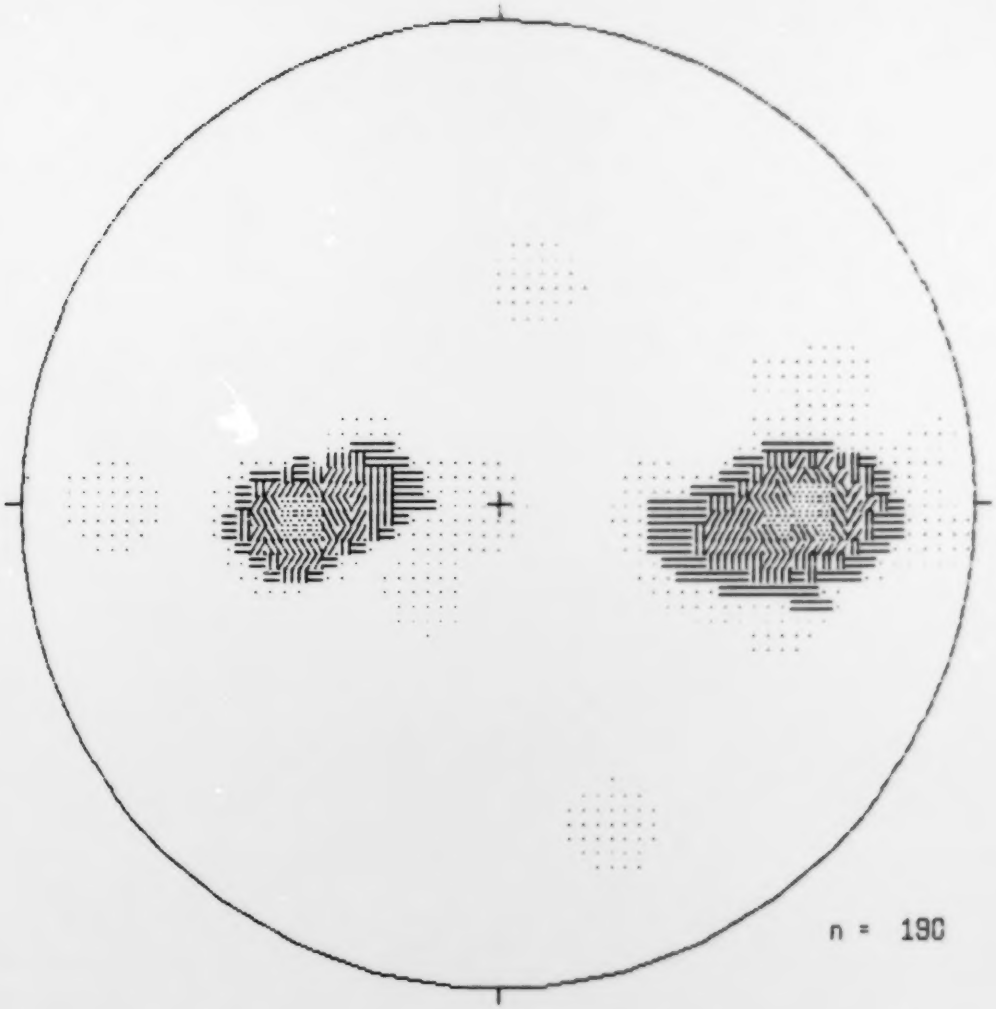
Dwg. No. 7

Trow

Belmoral Mine



0.5-4.5% 4.5-8.5% 8.5-12.5% 12.5-16.5%

16.5-20.5% 20.5-22.6%

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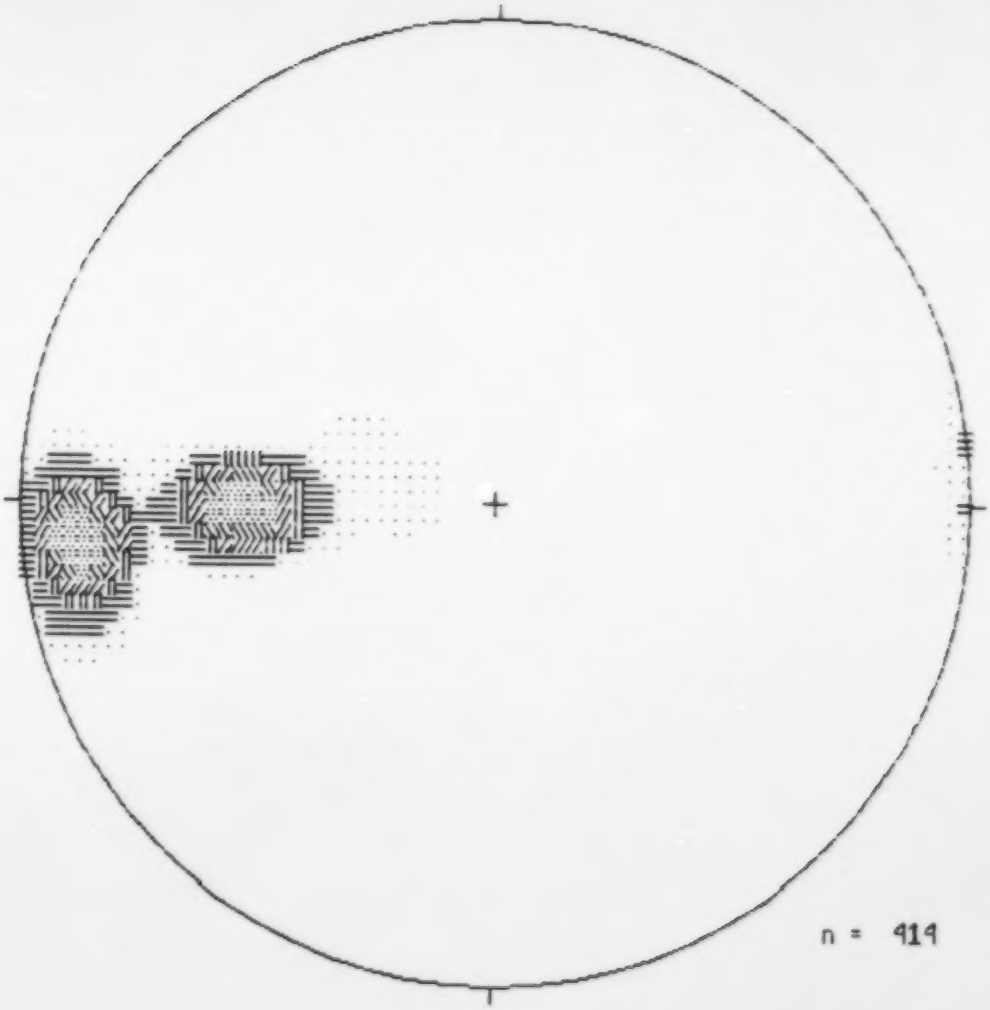
Dwg. No. 8

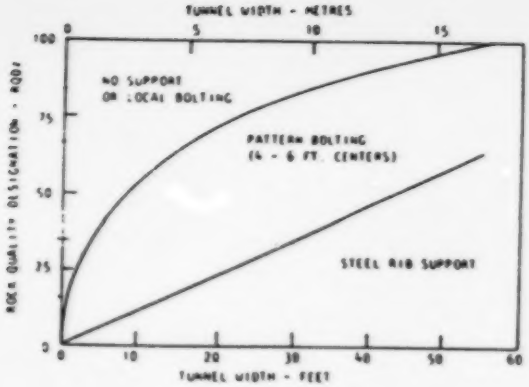
Trow

Chimo Mine

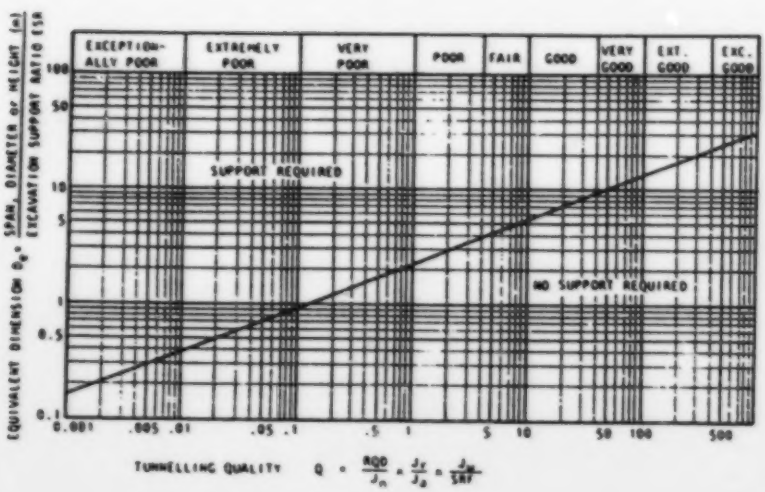


0.2-6.2% 6.2-12.2% 12.2-18.2% 18.2-24.2%

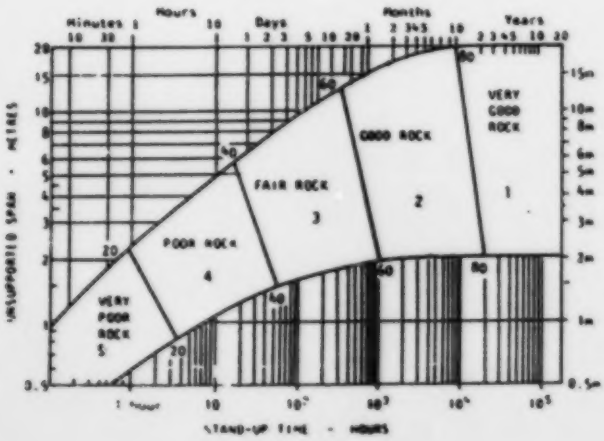
24.2-30.2% 30.2-37.0%



Merritt[6]



NGI[4]



CSIR - Bieniawski 5

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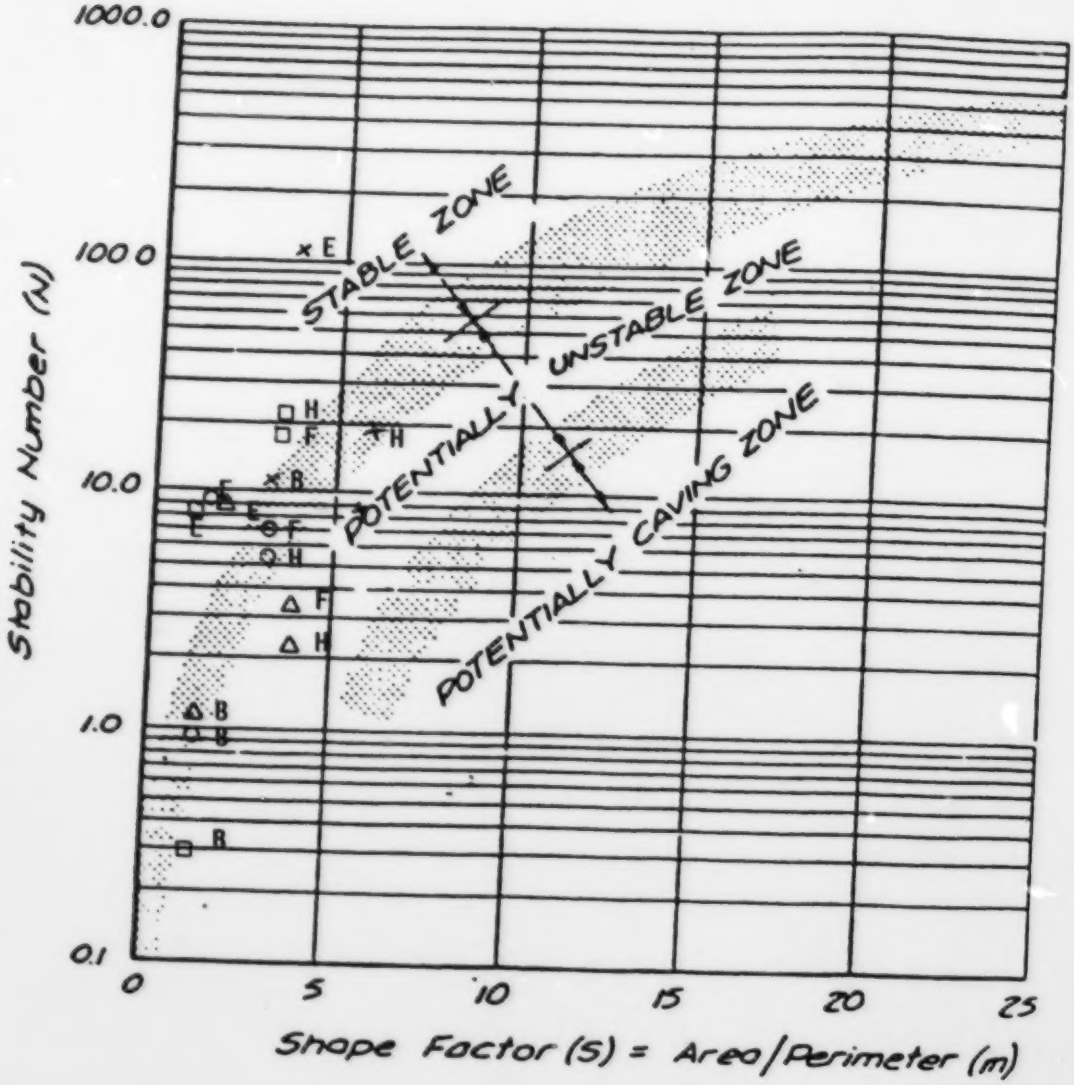
Dwg. No. 10

GRAPH OF STABILITY NUMBER (N)
VERSUS SHAPE FACTOR (S)

(after Mathews et Al. 1981)



× Pamour
△ Bousquet
○ Belmoral
□ Chimo

B - Back
E - Vertical End
H - Hangingwall
F - Footwall

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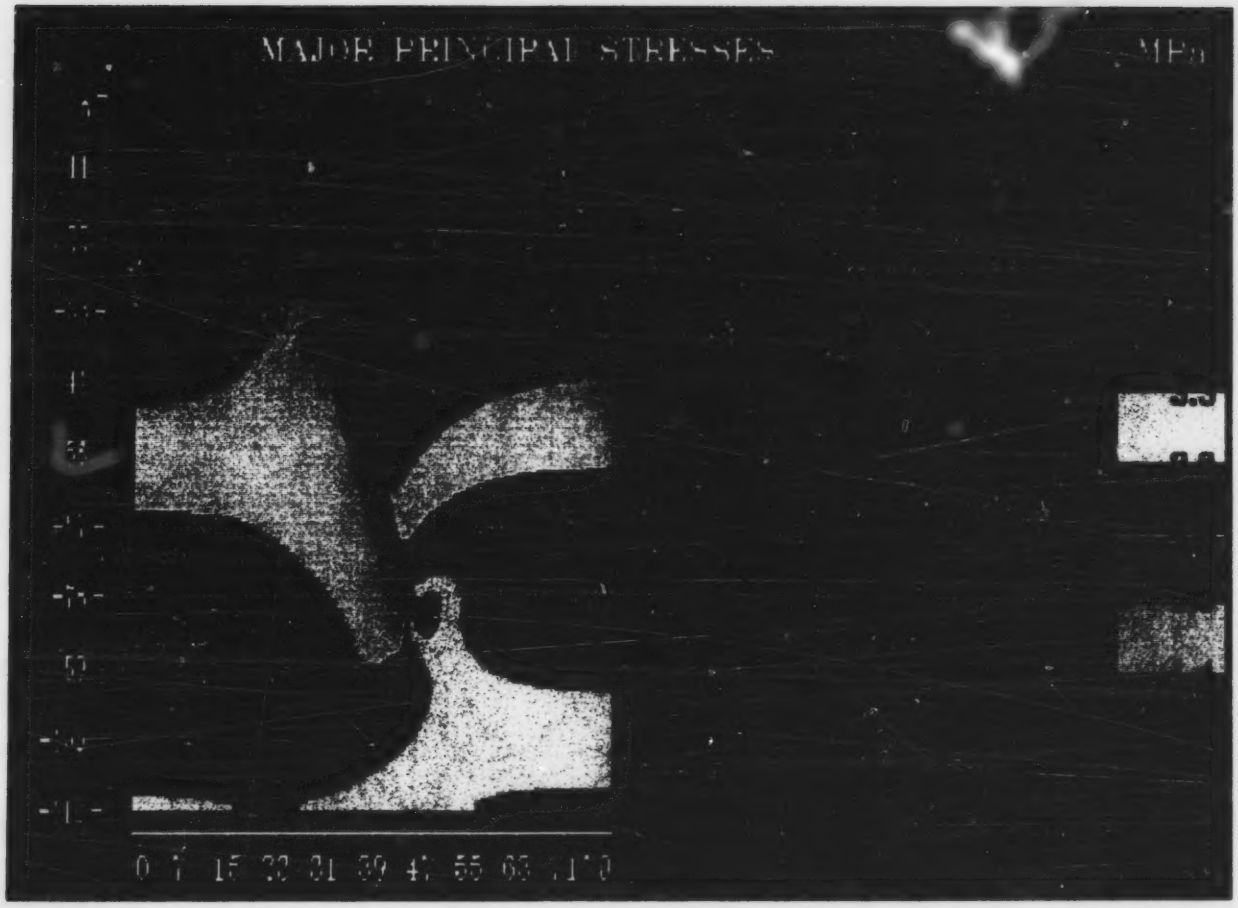
Dwg. No. 11

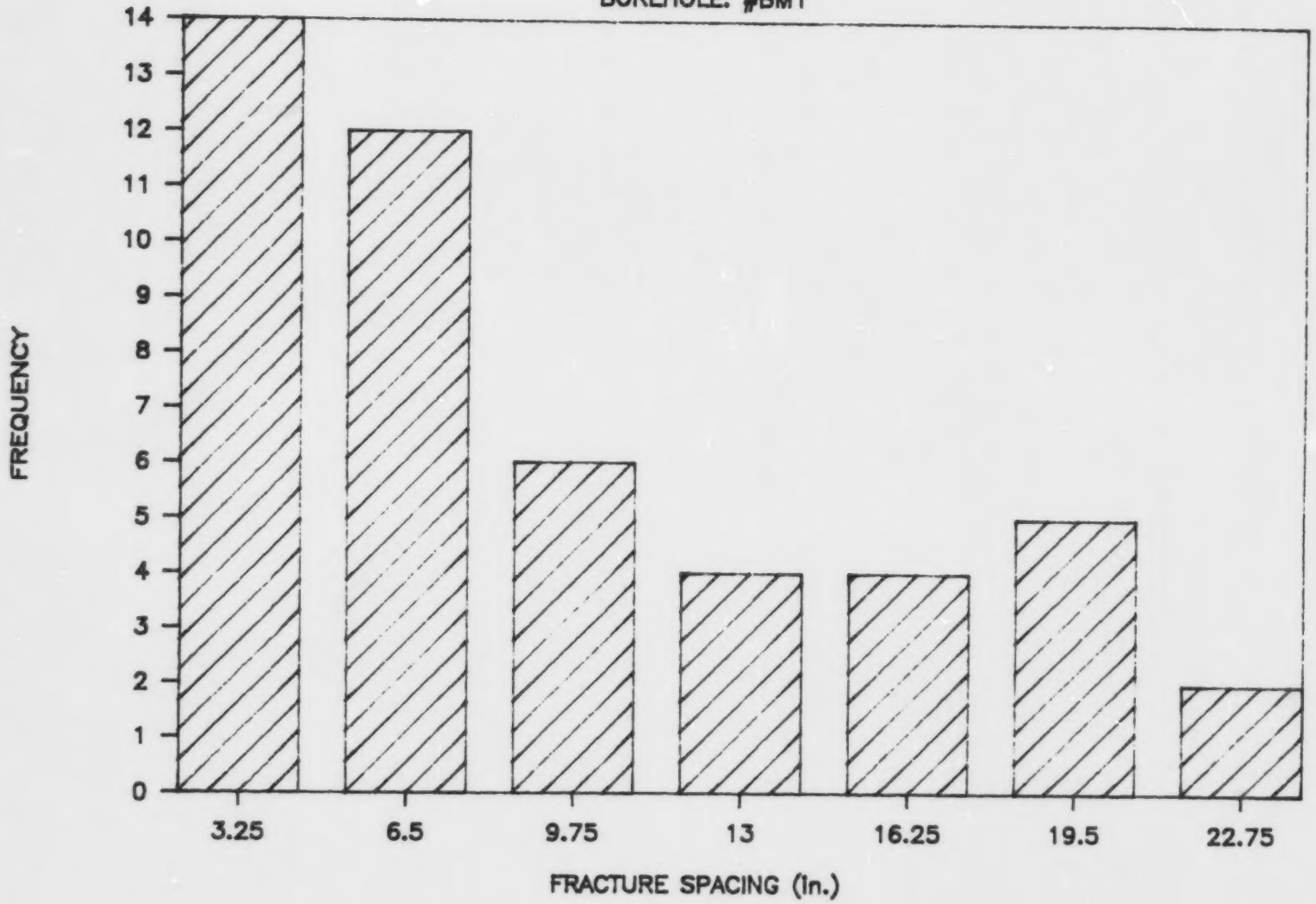


APPENDIX 1

Fracture Frequency Distributions

BELMORAL MINE

BOREHOLE: #BM1



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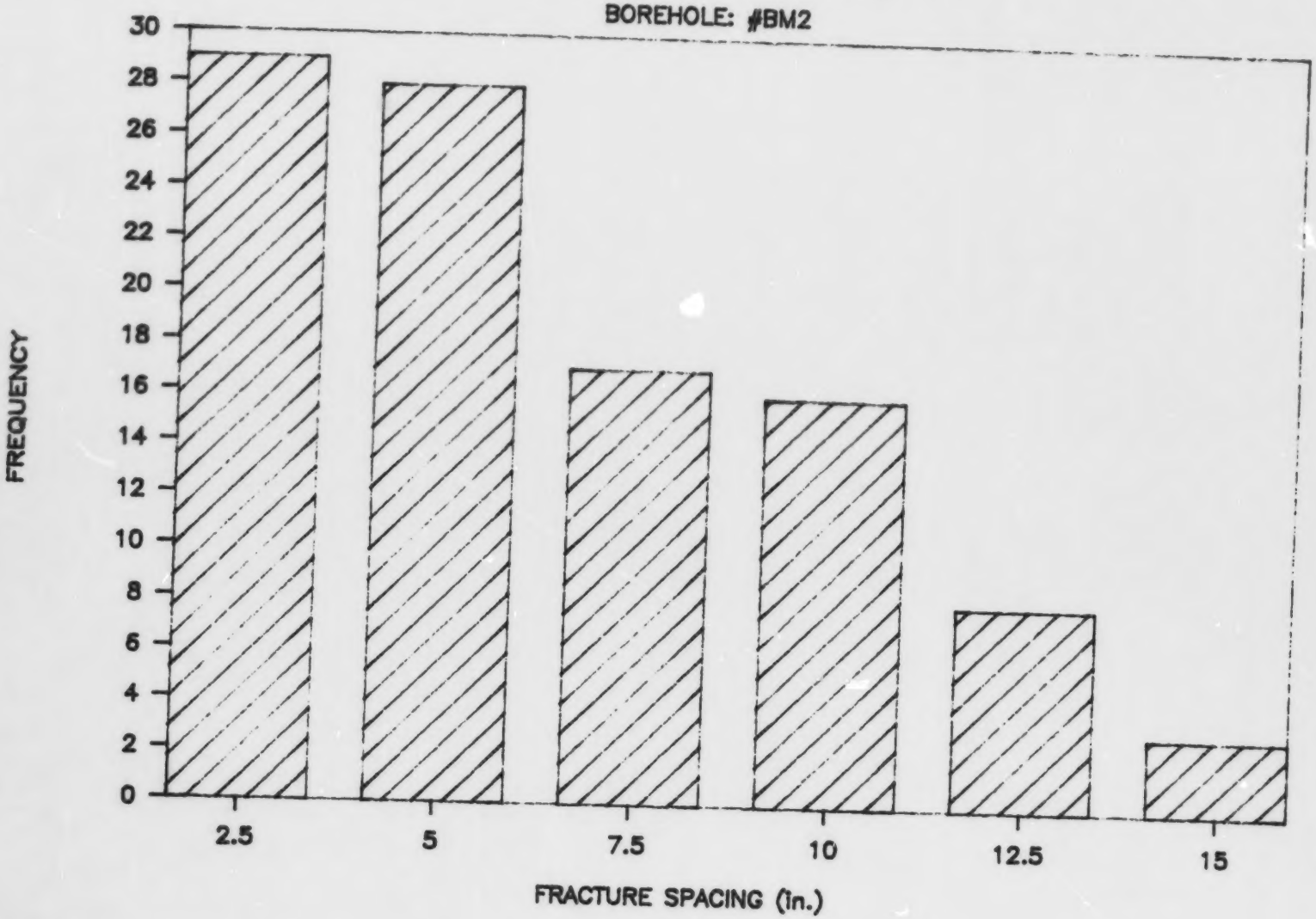
BELMORAL MINE

BOREHOLE: #BM2



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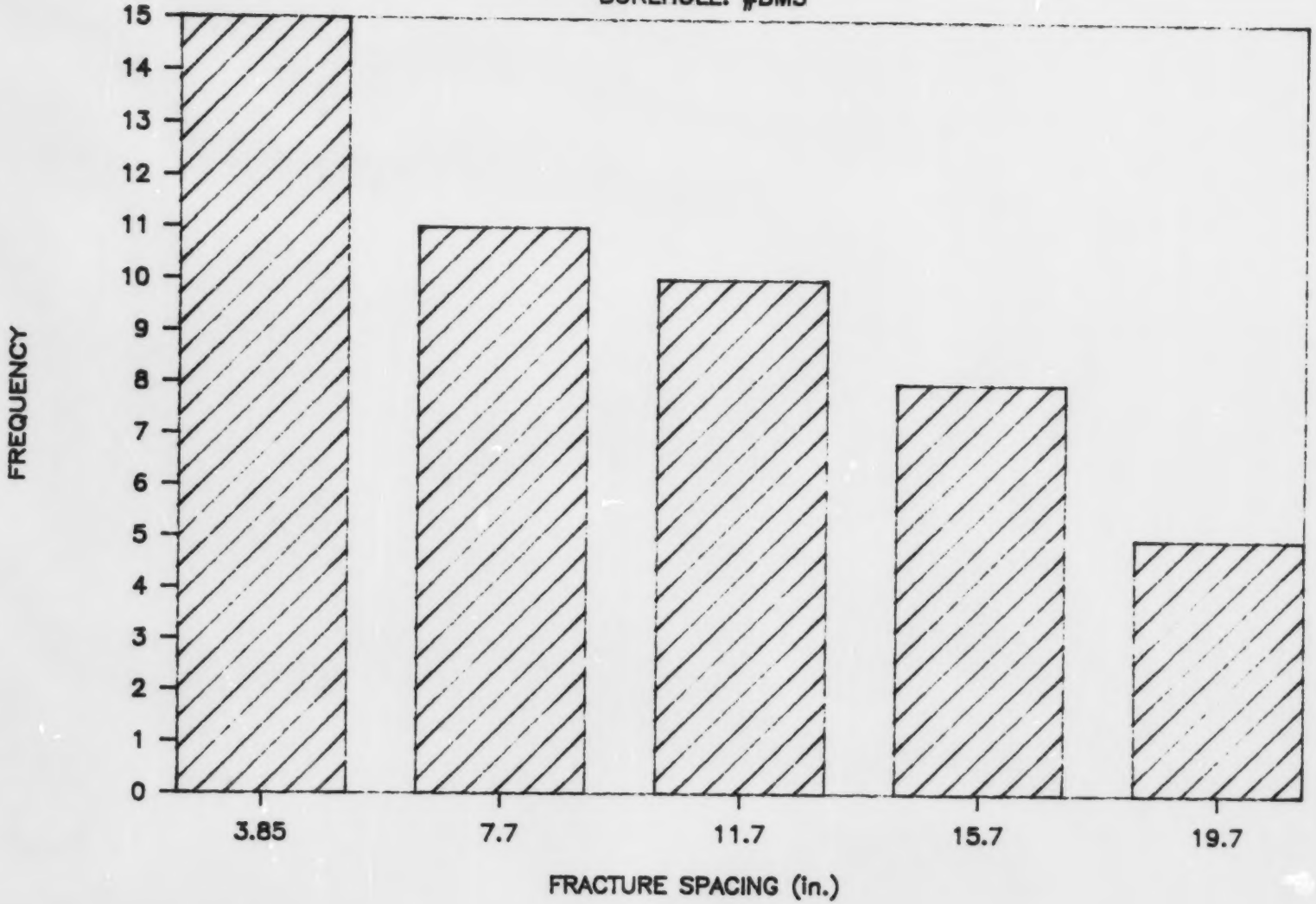
BELMORAL MINE

BOREHOLE: #BM3



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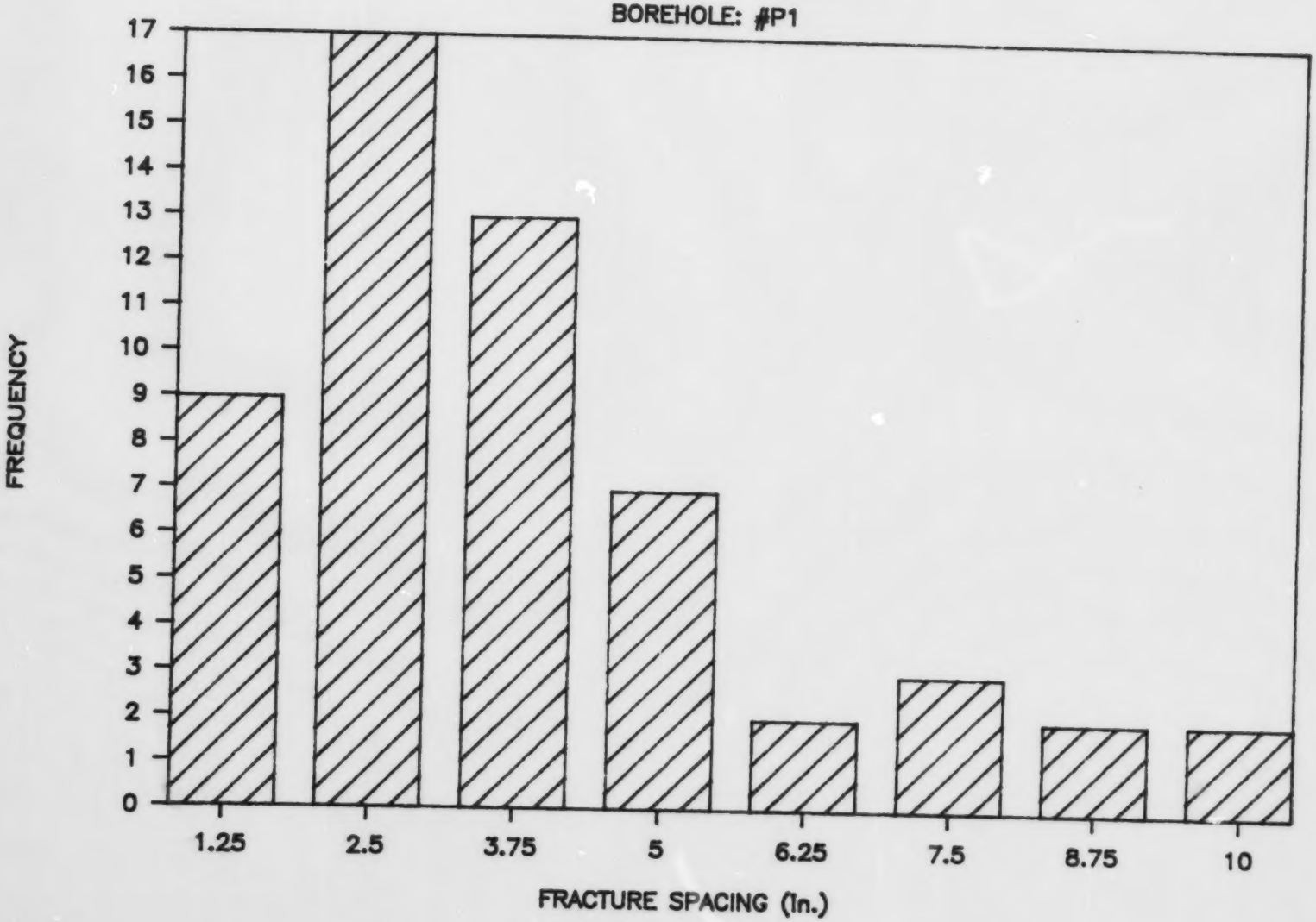
PAMOUR MINE

BOREHOLE: #P1



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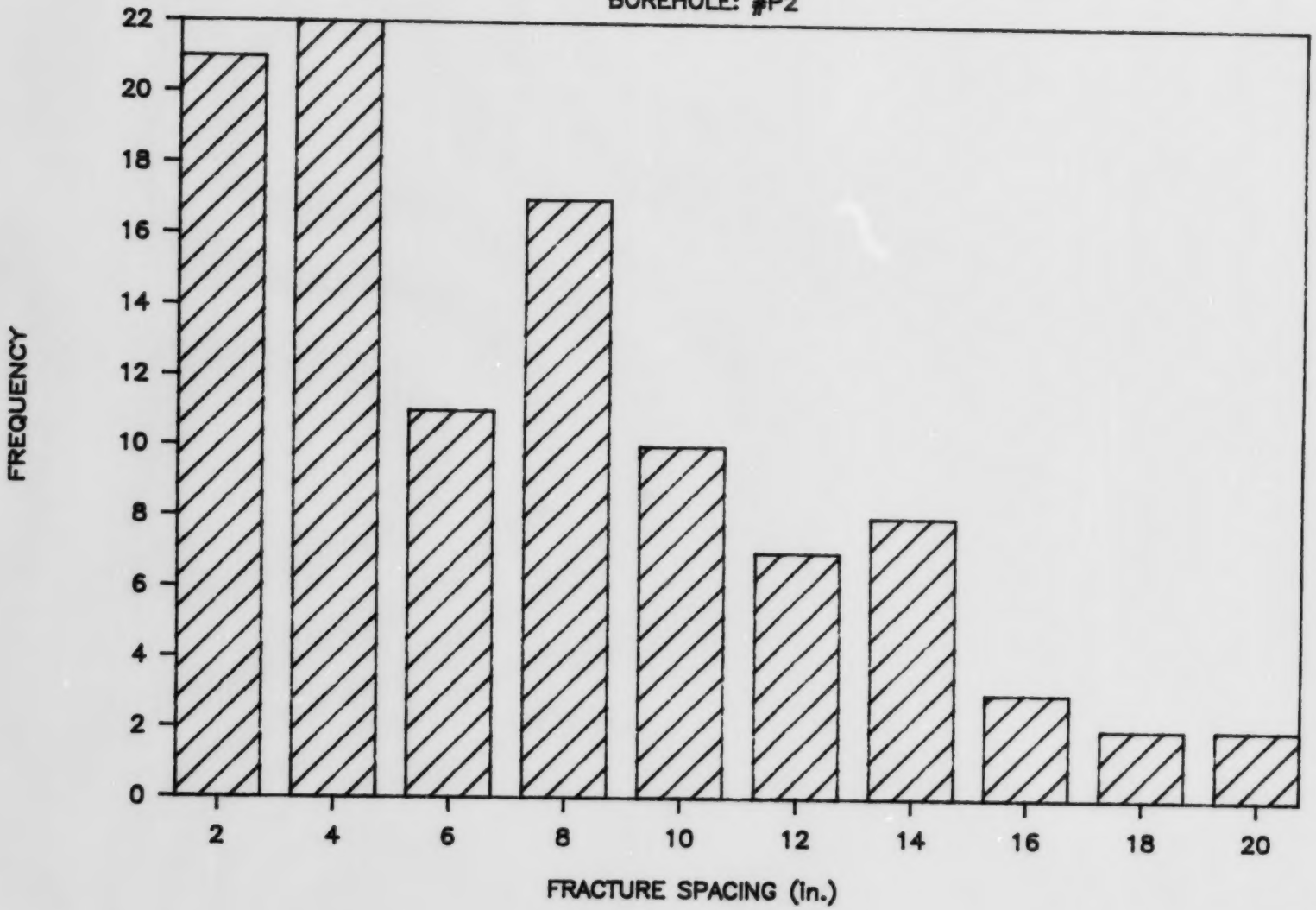
PAMOUR MINE

BOREHOLE: #P2



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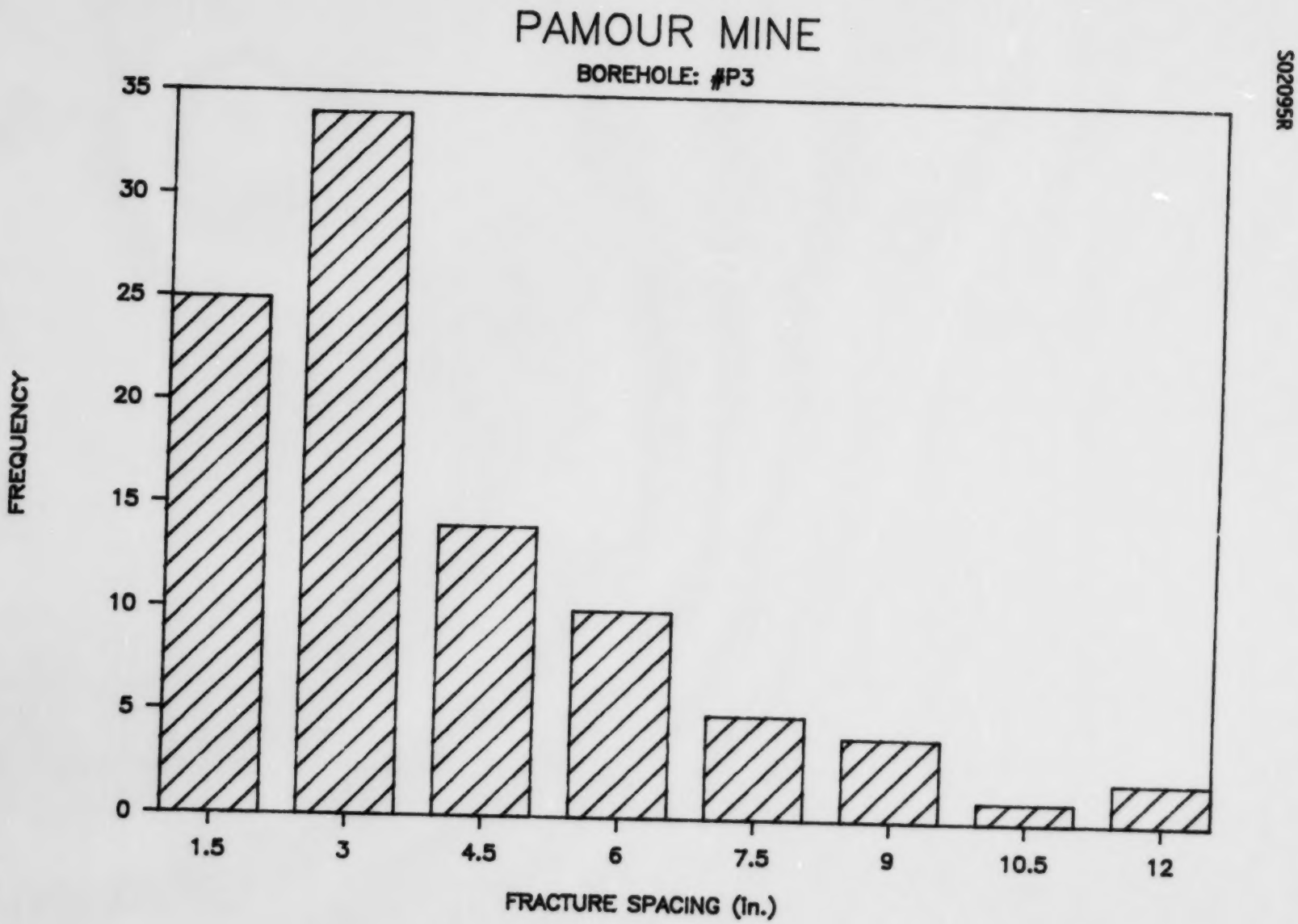
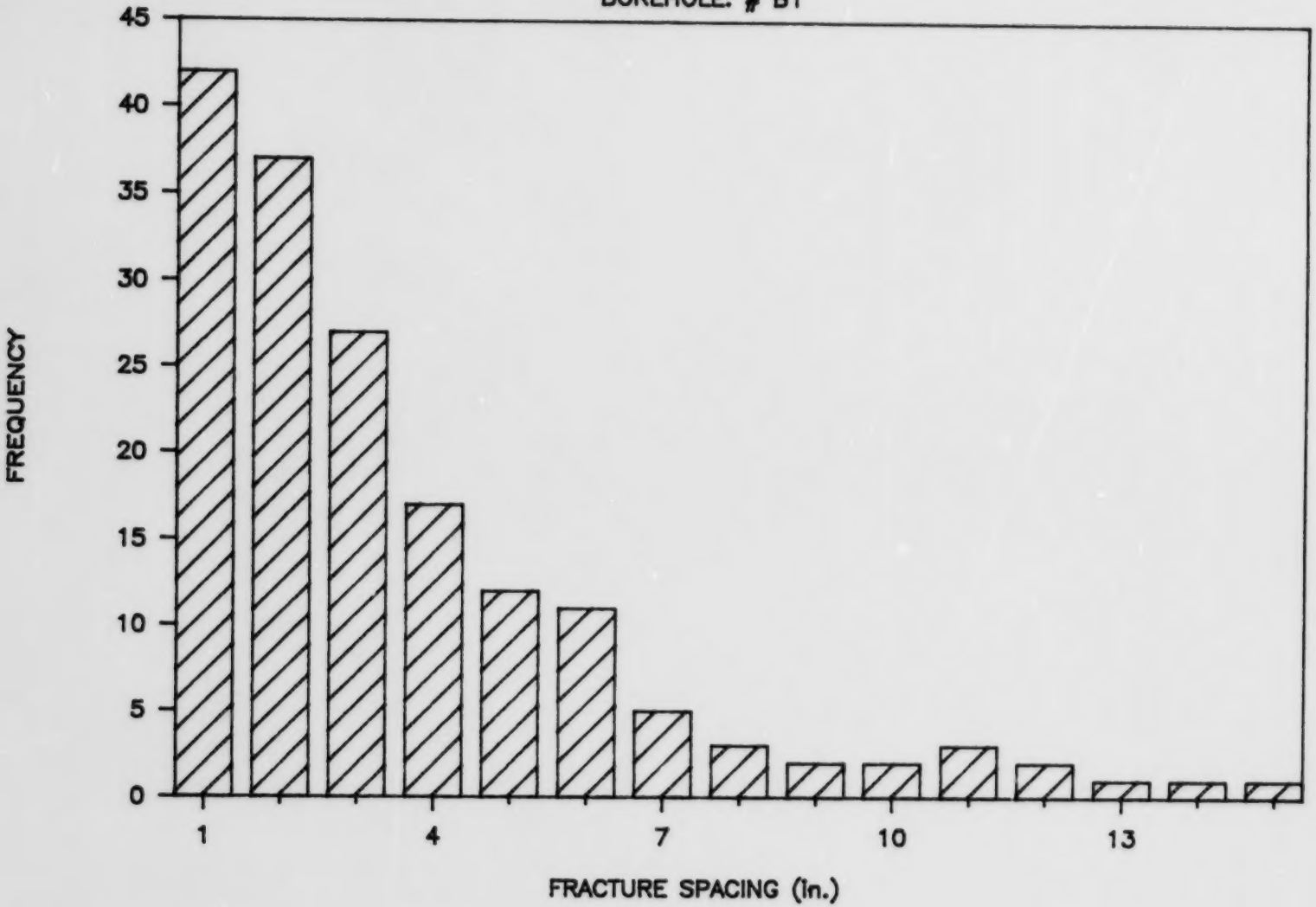

PAMOUR MINE
BOREHOLE: #P3
S02095R
FREQUENCY
35
30
25
20
15
10
5
0
1.5
3
4.5
6
7.5
9
10.5
12
FRACTURE SPACING (in.)

BOUSQUET MINE

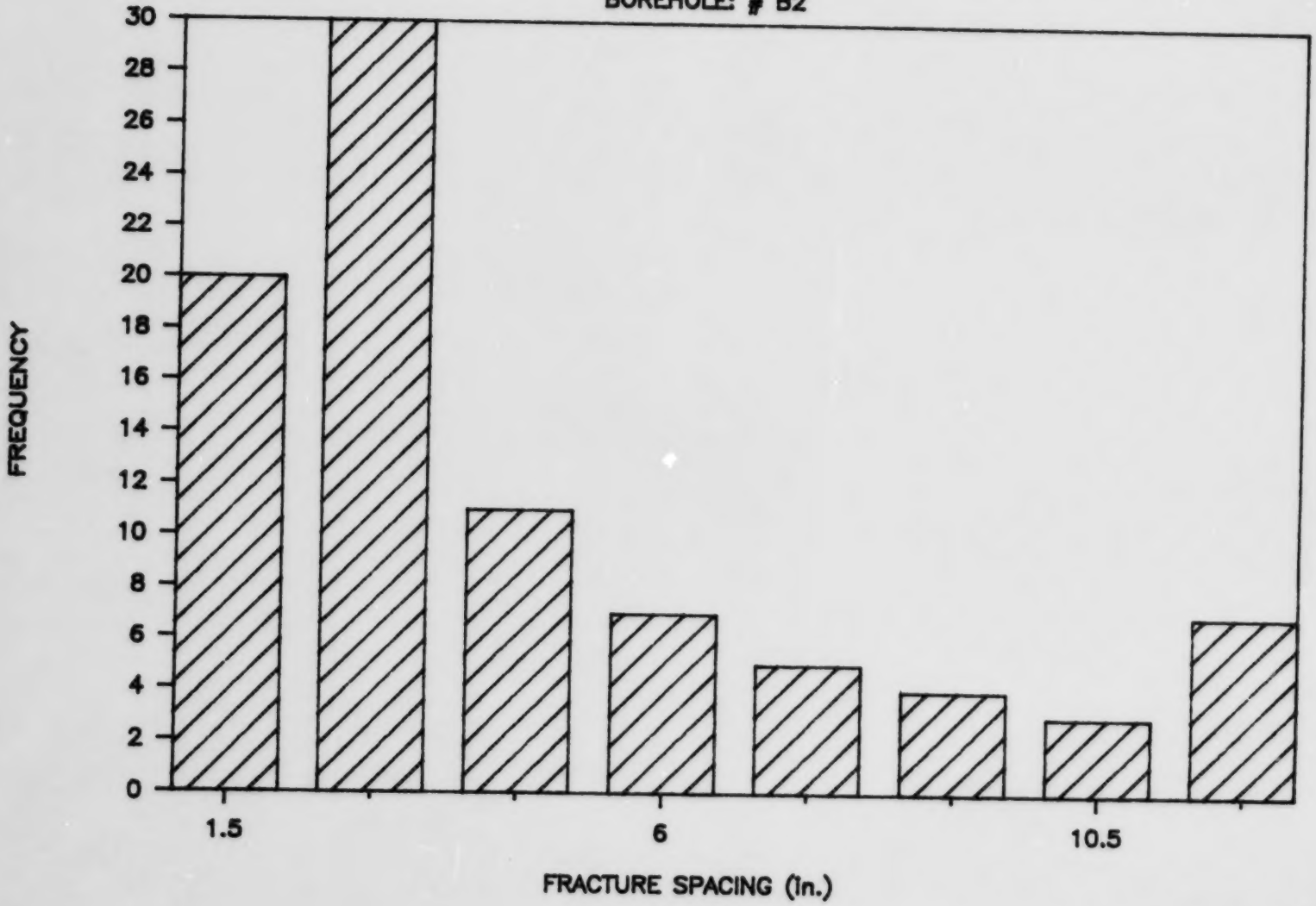
BOREHOLE: # B1



S02095R

BOUSQUET MINE

BOREHOLE: # B2



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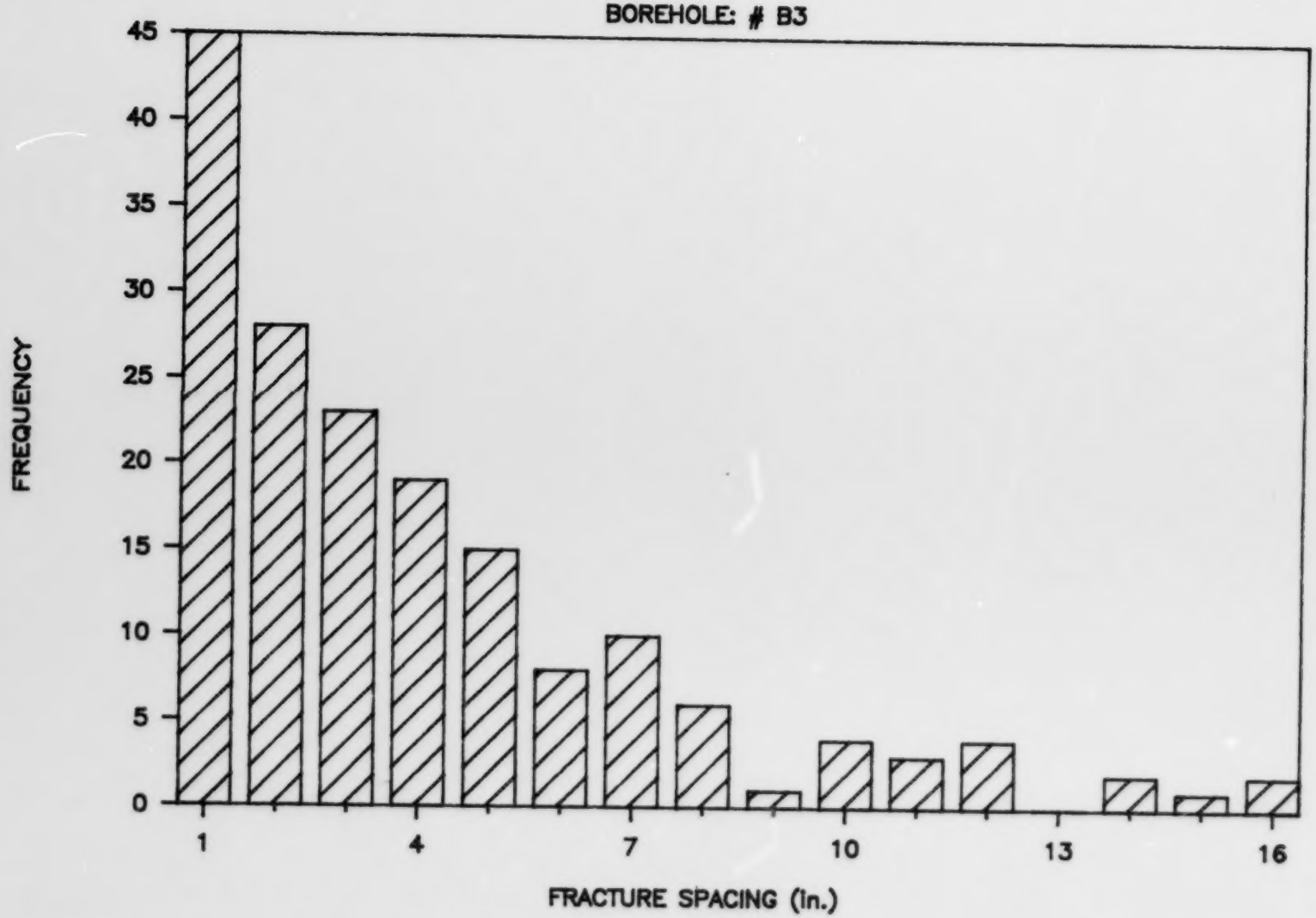
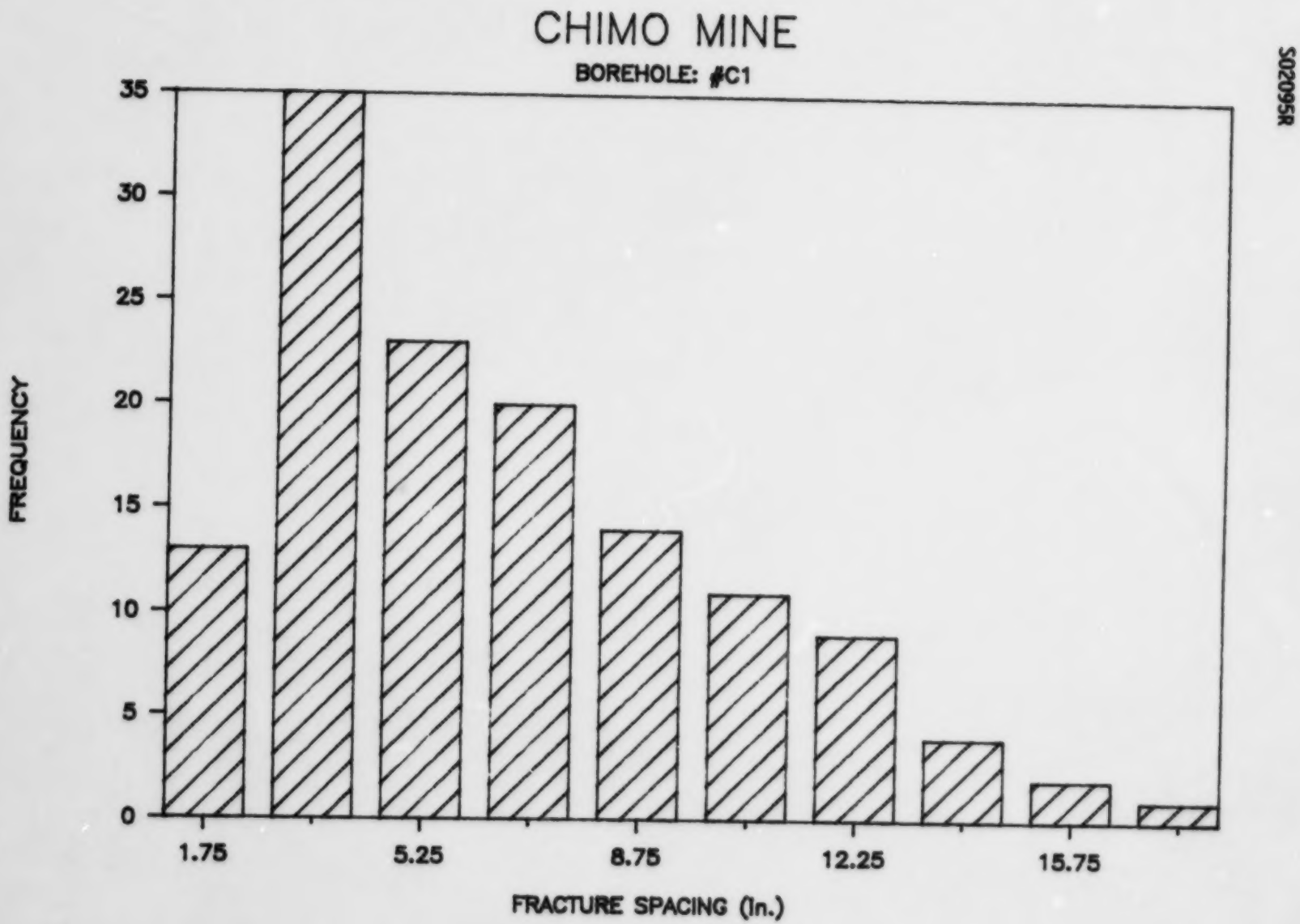
BOUSQUET MINE

BOREHOLE: # B3

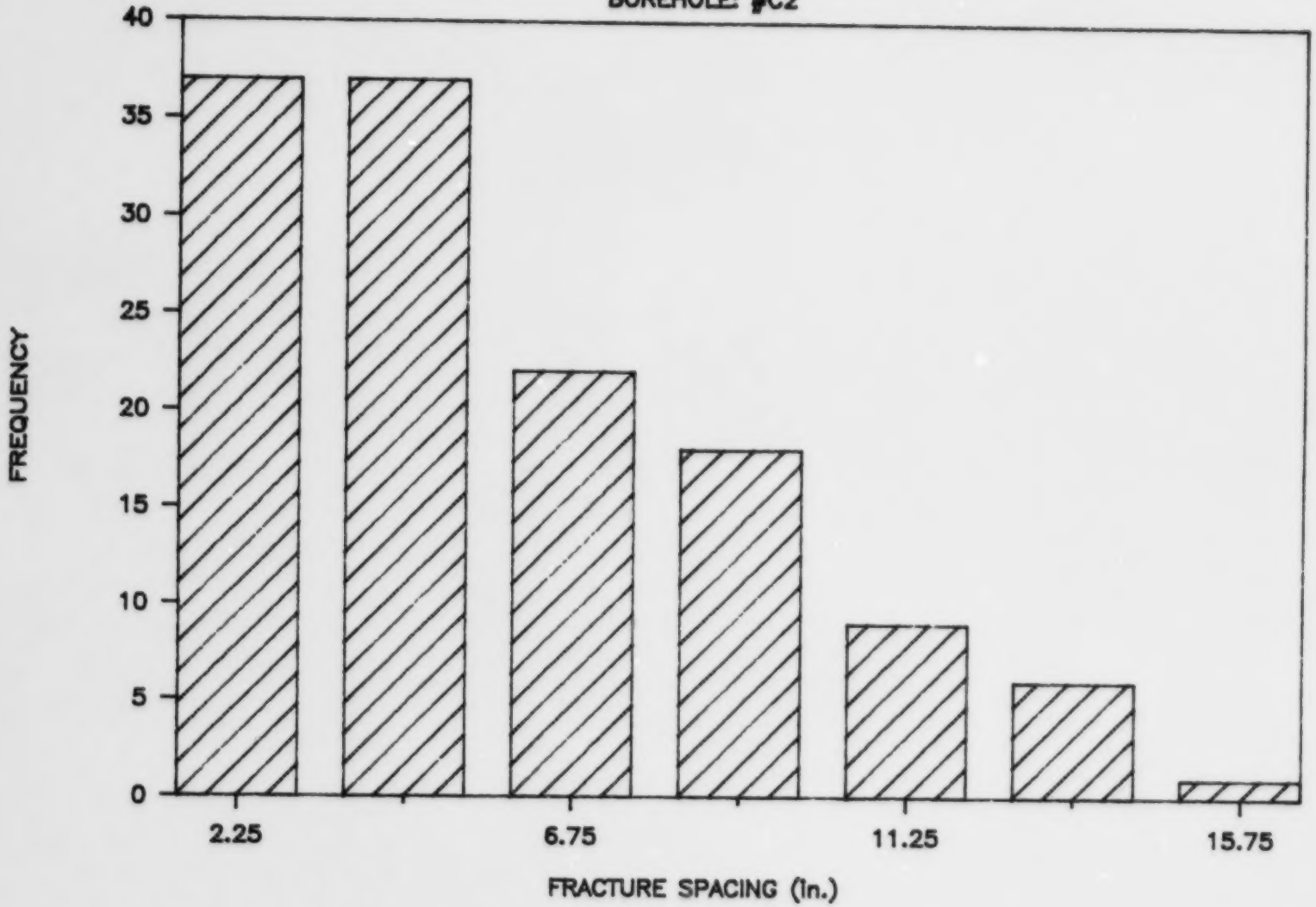


S02095R


CHIMO MINE
BOREHOLE: #C1
S02095R
FREQUENCY
35
30
25
20
15
10
5
0
1.75
5.25
8.75
12.25
15.75
FRACTURE SPACING (In.)

CHIMO MINE

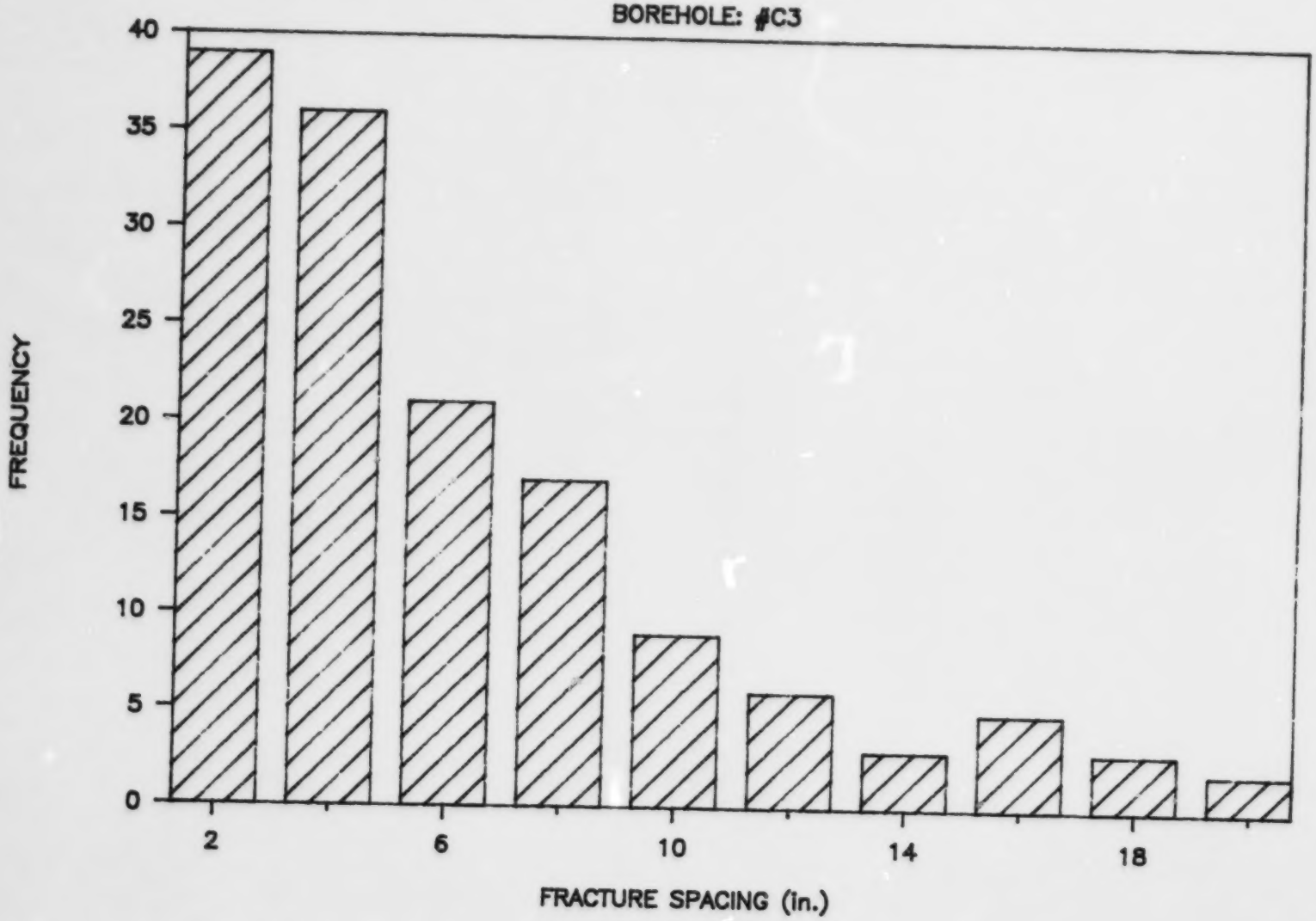
BOREHOLE: #C2



S020095R

CHIMO MINE

BOREHOLE: #C3



S02095R

Trow

APPENDIX 2

Core Logs

EXPLANATORY SHEET TO CORE LOG

COLUMN NO.	DESCRIPTION
1	Elevation of borehole collar.
2	Depth of geotechnical boundary in borehole.
3	Geological symbol for rock or soil material.
4	General description of geotechnical unit - qualitative description including rock type(s),percentage rock types,frequency and sizes of interbeds,colour,texture,weathering,strength,general joint spacing.
5-11	Joint (discontinuity) characteristics
5	Number of joint sets: a rock mass can be intersected by a number of joint sets of varying orientations.
6	Joint type: B = Bedding Joint C = Cross Joint
7	Orientation: only variations in dip can be identified in core; dip direction is obtained from field mapping or oriented core. F = Flat • 0 - 20° D = Dipping • 20 - 50° V = Vertical • 50 - 90°
8	Joint spacing: this is an approximate measure of spacing between joints in specific joint sets VW = Very Wide • >3 m W = Wide • 1 - 3 m M = Moderate • 30 cm - 1 m C = Close • 5 - 30 cm VC = Very Close • <5 cm
9	Roughness: RU = Rough Undulating RP = Rough Planar SU = Smooth Undulating SP = Smooth Planar LU = Slickensided Undulating LP = Slickensided Planar
10	Filling:

	Approx. ϕ_r
T = Tight,hard,non-softening	-
O = Oxidation surface staining only	25 - 35
SA = Slightly altered; clay-free	25 - 30
S = Sandy particles,clay-free	25 - 30
SI = Sandy and silty,minor clay	20 - 25
NC = Non softening clays,<5 mm	16 - 24
SC = Swelling clay fillings <5 mm	6 - 12

11	Aperture: estimated size of joint opening
12	Degree of weathering of rock material:

Unweathered	• no signs of discolouration or oxidation
Slightly weathered	• partial discolouration;fractures(joints) typically oxidized
Moderately weathered	• total discolouration
Highly weathered	• total discolouration;typically friable & pitted
Completely weathered	• resembles a soil;rock structure usually preserved

EXPLANATORY SHEET TO CORE LOG - Cont d

COLUMN NO.	DESCRIPTION
13.	Strength of rock material:

		Approx. Uniaxial Compressive Strength
Very high strength	• specimen can only be chipped by geological hammer	>200 MPa
High strength	• specimen requires a number of blows of geological hammer to fracture it;cannot be scraped with pocket knife	50 - 200 MPa
Medium strength	• specimen can be fractured by single firm blow of geological hammer;can be scraped with pocket knife, not peeled	15 - 50 MPa
Low strength	• shallow indentations made by firm blow with point of geological hammer;can be peeled by pocket knife with difficulty	4 - 15 MPa
Very low strength	• crumbles under firm blow with point of geological hammer; can be peeled by pocket knife	1 - 4 MPa

14 Fracture Frequency: Number of natural joints occurring over a metre length of core. All natural joints are counted irrespective of the number of joint sets.

Fracture frequency	Joint spacing
>0.3/m	Very wide • >3 m
0.3 - 1/m	Wide • 1 - 3 m
1 - 3/m	Moderate • 30 cm - 1 m
3 - 20/m	Close • 5 - 30 cm
<20/m	Very close • <5 cm

15 Run Number and Core Recovery:

(i) Drill run number;

(ii) Core Recovery is the total length of core pieces, irrespective of their individual lengths,obtained in a core run and expressed as a percentage of the length of that core run.

16 Rock Quality Designation (RQD): The total length of those pieces of sound core which are 10 cm or greater in length in a core run expressed as a percentage of the total length of that core run. Sound pieces of rock are those pieces separated by natural breaks and not machine breaks or subsequent artifical breaks.

RQD	Rock Mass Classification (After Deere)
0 - 25%	very poor
25 - 50%	poor
50 - 75%	fair
75 - 90%	good
90 - 100%	excellent

17 Core and Casing Sizes: changes of core and casing sizes are indicated

18 Water recovery, level and tests.

CORE LOG

Trow

Project	Orientation	Collar Elevation	Datum	Borehole No.
DSS Crown Pillar	-64°S	289'N.B.L.	Section 10	P1
Location	**Date Started**	**Completed**	**Logged By**	**Sheet** 1 of 1
Giant Yellowknife Pamour #1	07/09/88	18/09/88	S.McAuliffe	
Client	**Drilling Agency**	**Drill Type**	**Core Barrel & Bit Design**	**Project No.**
CANMET, EMR	Longyear	Longyear 34	NQ3-Face Ejection	S02095R

ELEVATION (ft)	DEPTH (ft)	SYMBOL	GENERAL DESCRIPTION	No. OF SETS	JOINT TYPE	ORIENTATION	SPACING	ROUGHNESS	FILLING	APERTURE (mm)	WEATHERING	STRENGTH	FRACTURE FREQUENCY	RUN No CORE RECOVERY %	RQD %	M/S	WATER RECOVERY, LEVEL & TESTS	Q/RMR
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
			Overburden casing to 30'															
29'6"			Conglomerate, mineralized zone, weathered except for slight weathering at joint surfaces between 34' to 36', joints typically are tight, close to very closely spaced with rough planar to rough undulated surfaces, a shear zone was intersected between 34' to 34' 11"	1	C	D	C/VC	RP/RU	T	0				#1 98.4	8.9		No Water Loss	1.8/62
														#2 84	53		No Water Loss	9.4/70
														#3 8.5	0		No Water Loss	-/-
														#4 42	0		No Water Loss	1.8/50
63'6"			Greywacke, footwall zone, rock mass is thinly bedded, joints are typically tight, closely spaced with smooth planar surfaces						S	30								
										0				#5 53	44.5		No Water Loss	2.6/42
									S	30								
										0								
74'6"			END OF BOREHOLE															

Remarks

CORE LOG

Trow

Project: DSS Crown Pillar	Orientation: -41°S	Collar Elevation: 321' N.B.L.	Datum: Section 34
Location: Giant Yellowknife Pasour #1	Date Started: 22/09/88	Completed: 24/09/88	Logged By: S. McAuliffe
Client: CANMET, EMR	Drilling Agency: Longyear	Drill Type: Longyear 34	Core Barrel & Bit Design: NQ3-Face Ejection

Borehole No.: P2

Sheet 1 of 1

Project No.: 902005R

ELEVATION (ft)	DEPTH (ft)	SYMBOL	GENERAL DESCRIPTION	Joint Characteristics: No. OF SETS	JOINT TYPE	ORIENTATION	SPACING	ROUGHNESS	FILLING	APERTURE (mm)	WEATHERING	STRENGTH	FRACTURE FREQUENCY	RUN No / CORE RECOVERY %	RQD %	M / S	WATER RECOVERY, LEVEL & TESTS	Q / RMR	
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	
			Overburden casing to 9'0"																
9'0"																			
			Greywacke, H.W. zone, tight, smooth planar, closely spaced joints	2	C	D	V/C	SP	T	O				#1 98	53	/	No water Loss	3.1 / 57	9'(
14'0"			Conglomerate, mineralized zone, joints are typically tight, close to moderately spaced with rough planar surfaces σ_c = 169.59 MPa σ_+ = 19.94 MPa E_{lab} = 65.0 GPa		C	V	-	SP	T	O									18'(
									S	3				#2 92	84	4.8 / 1.0	No water Loss	14.9 / 99	
														#3 95	95	4.8 / 1.0	No water Loss	38 / 92	26'(
				1	C	D	V/C	SP / RP	T	O				#4 100	100	4.8 / 1.0	No water Loss	26.7 / 84	32'(
														#5 100	96	4.8 / 1.0	No water Loss	25.6 / 92	42'(
														#6 100	92	4.8 / 1.0	No water Loss	18.4 / 82	52'(
														#7 100	90	4.8 / 1.0	No water Loss	12 / 82	62'(
72'0"								S	10										72'(
			END OF BOREHOLE																

Remarks

CORE LOG

Trow

Project: DSS Crown Pillar	Orientation: -52°S	Collar Elevation: 283' N.B.L.	Datum: Section 18	Borehole No.: P3
Location: Giant Yellowknife Pamour #1	Date Started: 19/09/88	Completed: 22/09/88	Logged By: S. McAuliffe	Sheet 1 of 1
Client: CANMET, EMR	Drilling Agency: Longyear	Drill Type: Longyear 34	Core Barrel & Bit Design: NQ3-Face Ejection	Project No.: S02095R

ELEVATION (ft)	DEPTH (ft)	SYMBOL	GENERAL DESCRIPTION	No. OF SETS	JOINT TYPE	ORIENTATION	SPACING	ROUGHNESS	FILLING	APERTURE	WEATHERING	STRENGTH	FRACTURE FREQUENCY	RUN No / CORE RECOVERY %	RQD %	M / S	WATER RECOVERY, LEVEL & TESTS	Q / RMR
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
			Overburden Rock and scrap fill material, casing to 6'															
														#1 1.6	0		Water loss at 9' Back at 10'	
17'			**Greywacke, H.W. zone, slight weathering between 17' and 26', joints are generally dipping and are close to very closely spaced, tight with smooth to rough planar surfaces** σ_c = 134.10 MPa σ_+ = ¯19.27 MPa E_{lab} = 44.20 GPa	1	C	D	C/RC	SP/RP	T	O				#2 20.3	43	2.58 / 1.0	Lost Water at 17'	17.2 / 55
														#3 70	30	2.58 / 1.0	No Water Return	2.7 / 49
										30				#4 93	40	2.58 / 1.0	No Water Return	2.4 / 57
47'			**Conglomerate, mineralized zone, tight, rough planar, closely spaced joints** σ_c = 85.23 MPa σ_+ = ¯19.27 MPa E_{lab} = 63.05 GPa	2	C	D	C	RP	T	O				#5 100	75	3.29 / 1.0	No Water Return	6.7 / 65
					C	D	M	RP	T	O				#6 65	82	3.29 / 1.0	No Water Return	16.4 / 94
61'			**END OF BOREHOLE** Broke through to stope															

6' 16' 26' 36' 46' 56' 61'

CORE LOG

Trow

Project: DSS Crown Pillar	Orientation: -76°N	Collar Elevation: 6318N	Datum: Section 7135
Location: Rouyquet Mine	Date Started: 30/09/88	Completed: 01/10/88	Logged By: L. Picicchia
Client: CANMET	Drilling Agency: Dominik	Drill Type: Insperation	Core Barrel & Bit Design: NQ3-Face Ejection

Borehole No.: B1

Sheet 1 of 1

Project No. S02095R

ELEVATION (ft)	DEPTH (ft)	SYMBOL	GENERAL DESCRIPTION	Joint Characteristics: No. OF SETS	JOINT TYPE	ORIENTATION	SPACING	ROUGHNESS	FILLING	APERTURE (mm)	WEATHERING	STRENGTH	FRACTURE FREQUENCY	RUN No / CORE RECOVERY %	RQD %	H / S	WATER RECOVERY, LEVEL & TESTS	Q / RMR	
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	
			Overburden Casing to 41'																
41'																			
46'			Porphyritic Tuff, H.W. zone, med. grey, 35-40% content of porphyroblasts σ_t = 6.05 MPa	1	B	V	C	SR/LP	T	O				#1 96	28.2	2.75/1.0	NO LOSS	1.4/36	41'
														#2 100	29	2.75/1.0	BLOCKING	2.5/36	45'
														#3 100	37.6	2.75/1	BLOCKED	2.8/36	47'
			Intermediate Tuff, F.W. zone, dark grey, 2-6% disseminated pyrite, sheared zone noted between 46.5' to 47', joints are typically vertical, closely spaced, tight surfaces are normally smooth planar with intermittent slickenside surfaces with chloritic and sericitic gauge											#4 93.7	44.5	2.75/1	NO WATER LOSS	3.2/41	51'
														#5 100	61	2.75/1	BLOCKED	3.1/41	57'
														#6 100	34.7	2.75/1	BLOCKED	2.5/35	60'
														#7 100	61.6	2.75/1	NO WATER LOSS	3.1/41	62'
														#8 100	84.5	2.75/1	NO WATER LOSS	4.1/45	67'
			σ_c = 29.95 MPa σ_t = 8.71 MPa E_{lab} = 63.35 GPa 045' E_{field} = 1.17 GPa 050' E_{field} = 2.62 GPa 055' E_{field} = 2.47 GPa 060' E_{field} = 0.86 GPa											#9 90	28	2.75/1	NO WATER	1.4/36	71'
														#10 100	66	2.75/1	NO WATER	3.5/41	75'
														#11 90	64.8	2.75/1	NO WATER	3.5/41	78'
87'														#12 100	53	2.75/1	NO WATER	2.7/41	82'
			END OF BOREHOLE																87'

Remarks

CORE LOG

Trow

Project: DSS Crown Pillar	Orientation: -72°N	Collar Elevation: 6317.2N	Datum: Section 7110	Borehole No.: B2
Location: Bousquet Mine	Date Started: 02/10/88	Completed: 03/10/88	Logged By: L. Piciacchia	Sheet 1 of 1
Client: CANMET	Drilling Agency: Dominik	Drill Type: Insperation	Core Barrel & Bit Design: NQ3-Face Ejection	Project No. S02095R

ELEVATION (ft)	DEPTH (ft)	SYMBOL	GENERAL DESCRIPTION	No. OF SETS	JOINT TYPE	ORIENTATION	SPACING	ROUGHNESS	FILLING	APERTURE (mm)	WEATHERING	STRENGTH	FRACTURE FREQUENCY	RUN No. CORE RECOVERY %	RQD %	M/S	WATER RECOVERY, LEVEL & TESTS	Q/RMR
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
			Overburden casing to 60'															
60'			Intermediate Tuff, H.W. banded with graphitic and chloritic bands .02" to .08" thick, poor recovery, blocky fill like material recovered		B	V	VC	O	T	O				#1 83	30		NO WATER RETURN	1.5/35
														#2 100	18.8		NO WATER RETURN	.44/30
			$\sigma_+ = {}^{-}9.32$ MPa	1				SP						#3 1	0		NO WATER RETURN	.5/30
								LP						#4 37	11		NO WATER RETURN	.55/30
91'			Mafic Tuff, ore zone, black quartz vein between 95.1' and 95.2' with some bands of intermediate tuffs, mafic tuffs contain 8-25% garnets, joints are typically related to bedding, vertical, close to very closely spaced and tight, smooth planar to slickenside planar											#5 47	0	3.63/1.0	NO WATER RETURN	.5/30
														#6 79	58.8	[illegible]	[illegible]	[illegible]
					NOT LOGGABLE									#7		3.63/1.0		
					C	F	M	O	T	O				#8 100	83	[illegible]	[illegible]	[illegible]
					B	V	C	SP	T	O				#9 89.8	75	3.63/1.0	NO WATER RETURN	3.8/44
			$\sigma_c = 67.08$ MPa	2														
					B	V	C	LP	T	O				#10 83	51	[illegible]	[illegible]	[illegible]
			$\sigma_+ = {}^{-}13.21$ MPa		C	F	M	RU	T	O				#11 100	95.8	3.63/1.0	NO WATER RETURN	4.8/47
			$E_{lab} = 57.54$ GPa		B	V	C	SP	T	O				#12 100	80	3.63/1.0	NO WATER RETURN	4.0/44
120'			END OF BOREHOLE															

Remarks

CORE LOG

Trow

Project: DSS Crown Pillar	Orientation: 81°N	Collar Elevation: 6276N	Datum: Section 7400
Location: Bousquet Mine	Date Started: 04/10/88	Completed: 05/10/88	Logged By: L. Piciacchia
Client: CANMET	Drilling Agency: Dominik	Drill Type: Insperation	Core Barrel & Bit Design: NQ3-Face Ejection

Borehole No.: B3

Sheet 1 of 1

Project No.: S02095R

ELEVATION (ft)	DEPTH (ft)	SYMBOL	GENERAL DESCRIPTION	Joint Characteristics: No. OF SETS	JOINT TYPE	ORIENTATION	SPACING	ROUGHNESS	FILLING	APERTURE (mm)	WEATHERING	STRENGTH	FRACTURE FREQUENCY	RUN No	CORE RECOVERY %	RQD %	M / S	WATER RECOVERY, LEVEL & TESTS	Q / RMR
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	
			Overburden casing to 24'																
24'			Rhyolitic Tuff, H.W. with numerous black quartz veins, 5-25% pyrite σ_c = 51.95 MPa σ_+ = 6.51 MPa E_{lab} = 56.75 GPa	1	B	V	VC	LP	T	O				#1 100	55	9.37/1.0	No water loss	3.8/47	24'
														#2 100	29	9.37/1.0	No water loss	2.5/42	27'
														#3 100	52	9.37/1.0	No water loss	2.6/47	29'
														#4 100	26.4	9.37/1.0	No water loss	3.5/42	33'
														#5 93	53.7	9.37/1.0	No water loss	2.6/47	36'
48'5"														#6 98	39	9.37/1.0	No water loss	1.45/42	41'
			Intermediate Tuff, F.W. 5-15% pyrite with sericitic zones, joints are of a bedding nature, vertical, very closely spaced, tight and slickenside planar σ_+ = 2.85 MPa											#7 100	56.9	9.37/1.0	No water loss	2.8/47	46'
														#8 100	46	9.37/1.0	No water loss	3.3/47	51'
														#9 100	90	9.37/1.0	No water loss	4.5/54	56'
														#10 100	58	9.37/1.0	No water loss	2.9/47	61'
														#11 100	82	9.37/1.0	No water loss	4.1/51	65'
														#12 100	42	9.37/1.0	No water loss	2.1/42	70'
														#13 100	60	9.37/1.0	No water loss	3/47	75'
														#14 100	45	9.37/1.0	No water loss	2.3/42	80'
90'			END OF BOREHOLE											#15 100	88	9.37/1.0	No water loss	4.4/51	85' 90'

Remarks

CORE LOG

Trow

Project: DSS Crown Pillar	Orientation: 90°	Collar Elevation: 10953N	Datum: Section 12700E
Location: Belmoral Mine	Date Started: 02/10/88	Completed: 05/10/88	Logged By: S. McAuliffe
Client: CANMET	Drilling Agency: Longyear	Drill Type: Longyear 34	Core Barrel & Bit Design:

Borehole No: BM1
Sheet 1 of 1
Project No. S02095R

ELEVATION (ft)	DEPTH (ft)	SYMBOL	GENERAL DESCRIPTION	No. OF SETS	JOINT TYPE	ORIENTATION	SPACING	ROUGHNESS	FILLING	APERTURE (mm)	WEATHERING	STRENGTH	FRACTURE FREQUENCY	RUN No / CORE RECOVERY %	RQD %	M / S	WATER RECOVERY, LEVEL & TESTS	Q / RMR
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
			Overburden casing to 23'															
23'																		23'
			Grano Diorite, H.W. zone, alternating between chloritic and hematitic alteration with blurred grains between slightly shear zones at 30-33' and 36-39'		C	D	VW	LU	T	O				#1 94	17.7	9.88 / 1.0	NO WATER LOSS	.7 / 38
					B	D	VW	LU	T	O				#2 97	40	9.88 / 1.0	LOST WATER RETURN	33' 1.6 / 35
			σ_t = ⁻9.20 MPa	4														39'
					C	V	m	SP	SA	O				#3 75	57	9.88 / 1.0	WATER RETURN CAME BACK AT 42'	2.3 / 40
50'					C	D	m	SP	T	O								
			Grano Diorite, F.W. zone, unaltered showing slight hematitic and chloritic alteration below 60', joints are typically dipping, moderately spaced with smooth planar to slickenside surfaces.		C	D	m	LU	T	O				#4 95.8	70.4	5.38 / 1.0	NO WATER LOSS	50' 5.3 / 52
				2	C	D	C	LU	T	O								60'
			σ_c = 70.30 MPa E_{lab} = 75.49 GPa											#5 95.4	56.3	5.38 / 1.0	NO WATER LOSS	4.2 / 52
69'			σ_t = ⁻16.15 MPa E_{field} = 10.77 GPa															69'
			END OF BOREHOLE															

Remarks

CORE LOG

Trow

Project: DSS Crown Pillar	Orientation: 69°N	Collar Elevation: 10958 N	Datum: Section 12800E
Location: Belmoral Mine	Date Started: 27/09/88	Completed: 02/10/88	Logged By: S. McAuliffe
Client: CANMET	Drilling Agency: Longyear	Drill Type: Longyear 34	Core Barrel & Bit Design: N Face Ejection

Borehole No: BM2

Sheet 1 of 1

Project No. S02095R

Elevation (ft)	Depth (ft)	Symbol	General Description	No. of Sets	Joint Type	Orientation	Spacing	Roughness	Filling	Aperture (mm)	Weathering	Strength	Fracture Frequency	Run No / Core Recovery %	RQD %	M / S	Water Recovery, Level & Tests	Q / RMR	Depth
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	
			Overburden casing to 15'																
15'																			15'
			Granodiorite, ore zone, altered and sheared joints, typically closely spaced, dipping, tight, slickenside planar σ_c = 77.4 MPa		C	V	C	LP	T	O				#1 100	76.6		NO WATER LOSS	1.3 / 55	20'
27'			E_{lab} = 70.20 GPa σ_+ = ⁻12.35 MPa	3	C	D	W	LP	T	O				#2 90	66.7		NO WATER LOSS	1.1 / 51	25'
			Granodiorite, F.W. side, altered with blurred grain textures, high content of felopars, weak to moderate hematized and chloritized alteration, joints are typically moderately to widely spaced, tight, slickenside planar and are typically dipping		C	D	C	LP	T	O				#3 93	67.9		NO WATER LOSS	1.1 / 51	30'
														#4 100	90	1064 / 1.0	NO WATER LOSS	1.5 / 58	35'
														#5 96.7	91.4	1064 / 1.0	NO WATER LOSS	1.5 / 58	40'
														#6 98	96.6	1064 / 1.0	NO WATER LOSS	1.6 / 58	45'
			σ_c = 77.44 MPa		C	F	M	LP	T	O				#7 99.2	80.7	1064 / 1.0	NO WATER LOSS	.81 / 55	
			σ_+ = ⁻11.98 MPa																55'
			E_{lab} = 65.88 GPa		C	D	C	LP	T	O									
				4										#8 95	76.3	1064 / 1.0	NO WATER LOSS	.76 / 55	
					C	D	W	LP	T	O									65'
71'					C	V	W	LP	T	O				#9 100	78.4	1064 / 1.0	NO WATER LOSS	.78 / 55	
			END OF BOREHOLE																71'9

Remarks

CORE LOG

Trow

Project: DSS Crown Pillar	Orientation: 90°	Collar Elevation: 10947 N	Datum: Section 12750E
Location: Belmoral Mine	Date Started: 06/10/88	Completed: 12/10/88	Logged By: S. McAuliffe
Client: CANMET	Drilling Agency: Longyear	Drill Type: Longyear 34	Core Barrel & Bit Design: N03-Face Ejection

Borehole No.: BM3

Sheet 1 of 1

Project No.: S02095R

ELEVATION (ft) 1	DEPTH (ft) 2	SYMBOL 3	GENERAL DESCRIPTION 4	No OF SETS 5	JOINT TYPE 6	ORIENTATION 7	SPACING 8	ROUGHNESS 9	FILLING 10	APERTURE (mm) 11	WEATHERING 12	STRENGTH 13	FRACTURE FREQUENCY 14	RUN No CORE RECOVERY % 15	RQD % 16	M / S 17	WATER RECOVERY LEVEL & TESTS 18	Q / RMR 19
			Overburden casing to 31'															
28'			Shear zone, ore zone, with some highly hematitic and sericitic alteration zones @40' E_{field} = 0.39 GPa	C	V	m	LP	T	O					#1 88	0		FULL WATER RETURN	.15 / 32
														#2 88	0		FULL WATER RETURN	.15 / 32
														#3 100	25		FULL WATER RETURN	.38 / 37
44'				C	V	m	SP	T	O					#4 100	0		FULL WATER RETURN	.15 / 32
			Grano Diorite, P.W. moderate hematitic alteration to 53' with blurred grains, sericitic alteration to 56', slight hematitic and sericitic alteration to 63'	S										#5 100	35		FULL WATER RETURN	.38 / 37
														#6 100	61		FULL WATER RETURN	1.3 / 54
				C	F	W	SP	T	O					#7 100	58		FULL WATER RETURN	1.16 / 54
			@45' E_{field} = 1.35 GPa σ_c = 76.2 MPa @50' E_{field} = 3.54 GPa σ_t = 14.68 MPa @55' E_{field} = 3.94 GPa E_{lab} = 72.49 GPa @60' E_{field} = 2.96 GPa											#8 100	70		FULL WATER RETURN	1.4 / 54
63'				C	V	C	LP/SP	T	O					#9 100	77		FULL WATER RETURN	1.5 / 58
			Grano Diorite, F.W. unaltered to 80', joints are typically crossed jointed, vertical, moderately spaced, smooth planar to alichensiide											#10 100	72	7.04 / 1.0	FULL WATER RETURN	1.4 / 54
														#11 100	80	7.04 / 1.0	FULL WATER RETURN	1.6 / 58
			σ_c = 81.4 MPa σ_t = 13.18 MPa E_{lab} = 65.36 GPa @65' E_{field} = 3.54 GPa	C	V	m	SP	T	O					#12 100	90	7.04 / 1.0	FULL WATER RETURN	1.8 / 61
80'														#13 100	89	7.04 / 1.0	LOST WATER	1.8 / 58
			END OF BOREHOLE															

28' 33' 37' 40'6' 43' 45' 48' 53' 58' 62' 67' 72' 77' 80'

CORE LOG

Trow

Project: DSS Crown Pillar	Orientation: 90°	Collar Elevation: 511.8m S.B.L.	Datum: Section 5+95E
Location: Chimo Mine	Date Started: 12/10/88	Completed: 14/10/88	Logged By: SM/LP
Client: CANMET	Drilling Agency: Dominik	Drill Type: Insperation	Core Barrel & Bit Design: NQ3-Face Ejection

Borehole No: C1

Sheet 1 of 1

Project No. S02095R

Elevation (ft)	Depth (ft)	Symbol	General Description	No. of Sets	Joint Type	Orientation	Spacing	Roughness	Filling	Aperture (mm)	Weathering	Strength	Fracture Frequency	Run No / Core Recovery %	RQD %	H/S	Water Recovery, Level & Tests	Q'/RMR	Depth (ft)
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	
			Overburden casing to 81'																
81'			Altered Tuff, F.W. chloritized amphiboles and calcified alterations between 100-121', 20% tuff, 80% andesite intrusion, joints are typically cross jointed, moderate to widely spaced, tight and smooth planar σ_c = 171.22 MPa σ_+ = 31.7 MPa E_{lab} = 89.05 GPa		B	D	M	SP	T	O				#1 100	85	7.41/1.0	No Water Loss	6.3/69	81'
					C	F	C	RP	T	O				#2 100	97	7.41/1.0	No Water Loss	7.2/72	91'
				3	C	V	W	SP	T	O				#3 100	43	7.41/1.0	No Water Loss	3.2/60	
														#4 96	65	7.41/1.0	No Water Loss	4.8/65	96'
														#5 100	45	7.41/1.0	No Water Loss	3.3/60	101'
														#6 100	83	7.41/1.0	No Water Loss	6.2/69	105
														#7 100	67	7.41/1.0	No Water Loss	5.0/65	110'
														#8 100	75	7.41/1.0	No Water Loss	5.6/69	115
121'			Massive Andesite, F.W. zone, only slight chloritic alteration σ_c = 165.23 MPa σ_+ = 43 MPa E_{lab} = 111.18 GPA											#9 100	37	2.84/1.0	No Water Loss	2.7/60	120'
														#10 100	75	2.84/1.0	No Water Loss	5.6/69	125
														#11 78	29	2.84/1.0	No Water Loss	2.3/60	130'
														#12 100	38	2.84/1.0	No Water Loss	2.8/60	133'
														#13 100	56	2.84/1.0	No Water Loss	4.2/65	138'
146'														#14 100	73	2.84/1.0	No Water Loss	5.4/65	141'
			END OF BOREHOLE																146'

Remarks

CORE LOG

Trow

Project: DSS Crown Pillar	Orientation: -77°	Collar Elevation: 513m S.B.L.	Datum: Section 6+60E
Location: Chimo Mine	Date Started: 08/10/88	Completed: 09/10/88	Logged By: SM/LP
Client: CANMET	Drilling Agency: Dominik	Drill Type: Insperation	Core Barrel & Bit Design: NQ3-Face Ejection

Borehole No: C2

Sheet 1 of 1

Project No: S02095R

ELEVATION (m)	DEPTH (m)	SYMBOL	GENERAL DESCRIPTION	No OF SETS	JOINT TYPE	ORIENTATION	SPACING	ROUGHNESS	FILLING	APERTURE (mm)	WEATHERING	STRENGTH	FRACTURE FREQUENCY	RUN No / CORE RECOVERY (%)	RQD (%)	M / S	WATER RECOVERY LEVEL & TESTS	Q / RMR
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
			Overburden casing to 106'															
106'			Andesite, H.W. chloritized, alterations of amphiboles between 114'-157', calcified alterations between 106'-114' and 157'-160', numerous quartz veins in bedding direction in both alteration forms σ_c = 105.92 MPa σ_- = 17.97 MPa E_{lab} = 52.58 GPa	2	B	D	C	SP	T	O				#1 100	21	4.53/1.0	NO WATER LOSS	3.5/55
														#2 100	29	4.53/1.0	NO WATER LOSS	4.8/60
														#3 100	82	4.53/1.0	NO WATER LOSS	13.7/69
														#4 100	45	4.53/1.0	NO WATER LOSS	7.5/60
					C	F	VC	RP	T	O				#5 100	23	4.53/1.0	NO WATER LOSS	3.8/55
														#6 100	68	4.53/1.0	NO WATER LOSS	11.3/65
														#7 100	21	4.53/1.0	NO WATER LOSS	3.5/55
														#8 81	27	4.53/1.0	NO WATER LOSS	4.5/60
				3	B	D	C	SP/LP	T/RP	1				#9 100	67	4.53/1.0	NO WATER LOSS	6.0/65
					C	F	M	RP	T	O				#10 100	52	4.53/1.0	NO WATER LOSS	4.6/65
					C	V	VW	SP/RP	T	O				#11 90	35	4.53/1.0	NO WATER LOSS	3.1/60
160'														#12 100	66	4.53/1.0	NO WATER LOSS	5.9/65
			Andesite, ore zone, chloritic and calcitic alteration throughout with distinct graphitic seams along bedding											#13 100	33	4.53/1.0	NO WATER LOSS	2.9/60
166'														#14 100	0	4.53/1.0	NO WATER LOSS	.89/55
			END OF BOREHOLE															

Joint Characteristics

Depth scale: 106', 111', 115', 120', 125', 129', 134', 137', 141', 146', 150', 155', 160', 163', 166'

Remarks:

CORE LOG

Trow

Project: DSS Crown Pillar	Orientation: -70°S	Collar Elevation: 511.8m S.B.L.	Datum: Section 6+97.5E	Borehole No: C3
Location: Chimo Mine	Date Started: 06/10/88	Completed: 07/10/88	Logged By: SM/LP	Sheet 1 of 1
Client: CANMET	Drilling Agency: Dominik	Drill Type: Insperation	Core Barrel & Bit Design: NQ3-Face Ejection	Project No: S02095R

ELEVATION (m)	DEPTH (m)	SYMBOL	GENERAL DESCRIPTION	No OF SETS	JOINT TYPE	ORIENTATION	SPACING	ROUGHNESS	FILLING	APERTURE (mm)	WEATHERING	STRENGTH	FRACTURE FREQUENCY	RUN/CORE RECOVERY	RQD	M/S	WATER RECOVERY LEVEL & TESTS	Q/RMR
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
			Overburden casing to 116'															
116'			Andesite, H.W. with chloritized alteration of amphiboles between 116'-134' and 152'-169', calcified alteration between 134'-152' and 169'-181', numerous quartz veins in clacified region 5-15% arseno pyrite below 169', predominate joint set due to bedding weakness, dipping, closely spaced, smooth planar σ_c = 87.48 MPa σ_+ = $^-$15.63 MPa E_{lab} = 59.06 GPa Altered m = 3.08 s = 1.0 Intact m = 4.87 s = 1.0	2	B	D	C	SP	T	O				#1 100	85	N	NO WATER LOSS	4.2/69
														#2 83	80		NO WATER LOSS	13.3/69
														#3 100	75		NO WATER LOSS	12.5/65
					C	F	M	RP	T	O				#4 100	51		NO WATER LOSS	8.5/65
														#5 100	58		NO WATER LOSS	4.7/65
														#6 100	33		NO WATER LOSS	5.5/60
														#7 100	85		NO WATER LOSS	14.2/69
														#8 96	58		NO WATER LOSS	4.7/65
														#9 100	80		NO WATER LOSS	13.3/69
														#10 100	66		NO WATER LOSS	11/65
														#11 100	66		NO WATER LOSS	11/65
														#12 100	83		NO WATER LOSS	13.8/69
														#13 100	47		NO WATER LOSS	7.8/60
														#14 100	11		NO WATER LOSS	1.8/55
181'														#15 100	52		NO WATER LOSS	8.7/65
			END OF BOREHOLE															

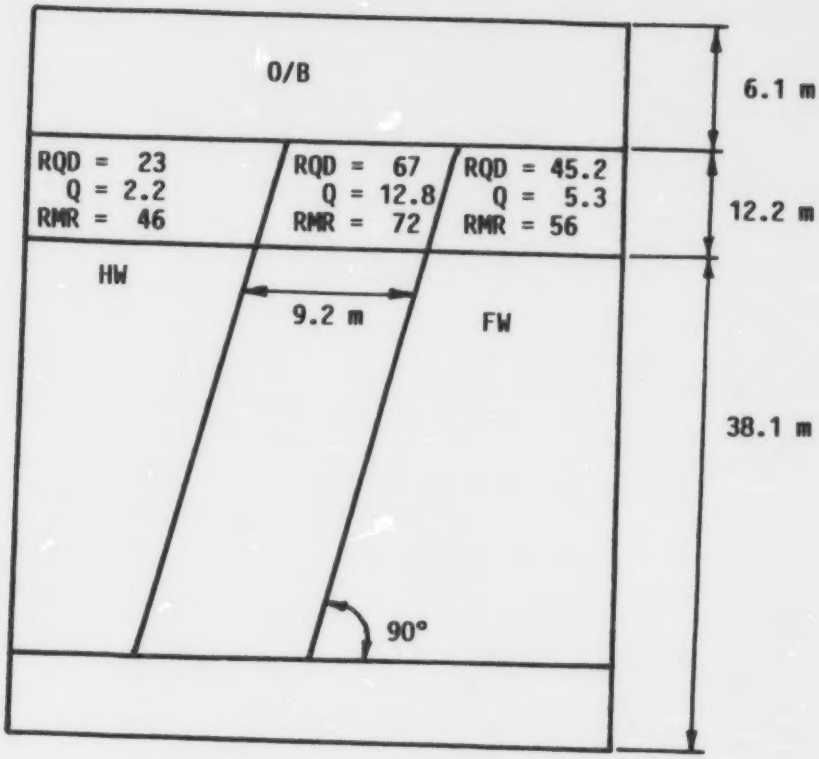
Remarks

APPENDIX 3

Simple Empirical Models

Trow

Pamour Mine



Merritt Support System

For a 9.2 m span, no support required for RQD > 80% and pattern bolting for RQD between 30% and 80%. Pattern bolting required.

NGI Tunneling Quality

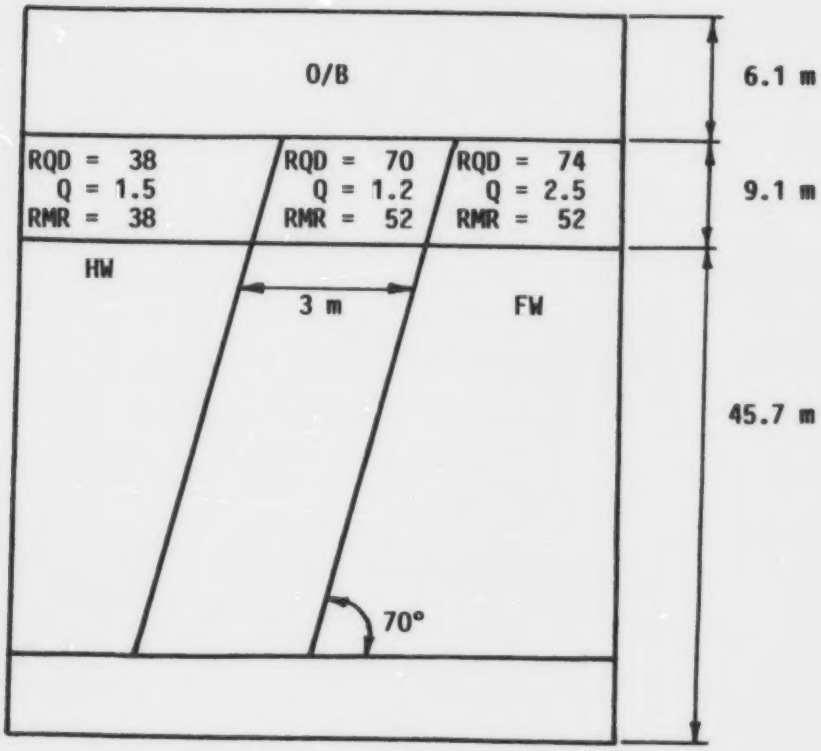
For a 9.2 m span and an excavation support ratio of 3.0 which implies that De = 3.07. This implies no support required for Q values > 2. No support required.

CSIR Standup Time

For a 9.2 m opening with RMR = 65, yields a standup time of 2½ months. Unreasonable standup time.

Trow

Belmoral Mine



Merritt Support System

For a 3.05 m span, no support is required for an RQD > 50% and pattern bolting needed for between 5% and 50% RQD. Pattern bolting on H/W only.

NGI Tunneling Quality

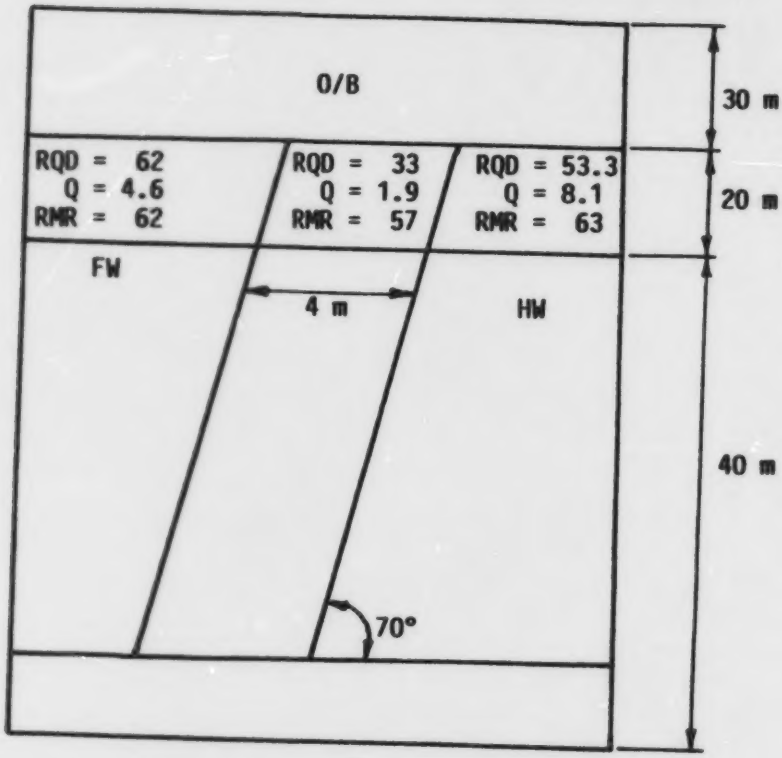
For a 3.05 m span and ESR = 3; De = 1 implies that no support needed for Q > 0.1. No support

CSIR Standup Time

For a 3.05 m wide opening and RMR = 50 implies standup time of 10 days. Not reasonable.

Trow

Chimo Mine



Merritt Support System

For a 4 m span, no support is required for RQD's > 55% and pattern bolting is required between 10 and 55%. No support.

NGI Tunneling Quality

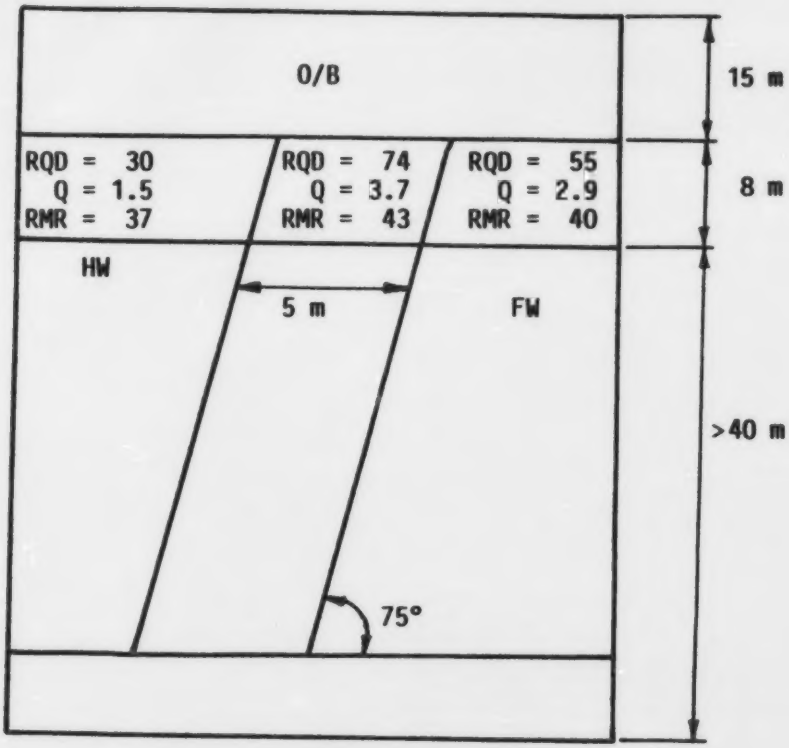
For a 4 m span and ESR = 3; De = 1.33 implies no support needed for Q > 0.3. No bolting.

CSIR Standup Time

For a 4 m opening and RMR = 62% implies that standup time = 2½ months. Not reasonable.

Trow

Bousquet Mine



Merritt Support System

For a 5 m span, no support required for RQD >70%, pattern bolting for RQD between 15% and 70%. Pattern bolting on HW/FW only.

NGI Tunneling Quality

For a 5 m span and excavation support ratio of 3.0 implies that De = 1.7. This implies no support required for Q>0.3. No support required.

CSIR Standup Time

For a 5 m span, an RMR = 40 gives standup time of less than 1 day. Unreasonable prediction.

Trow

APPENDIX 4

Mathews Method

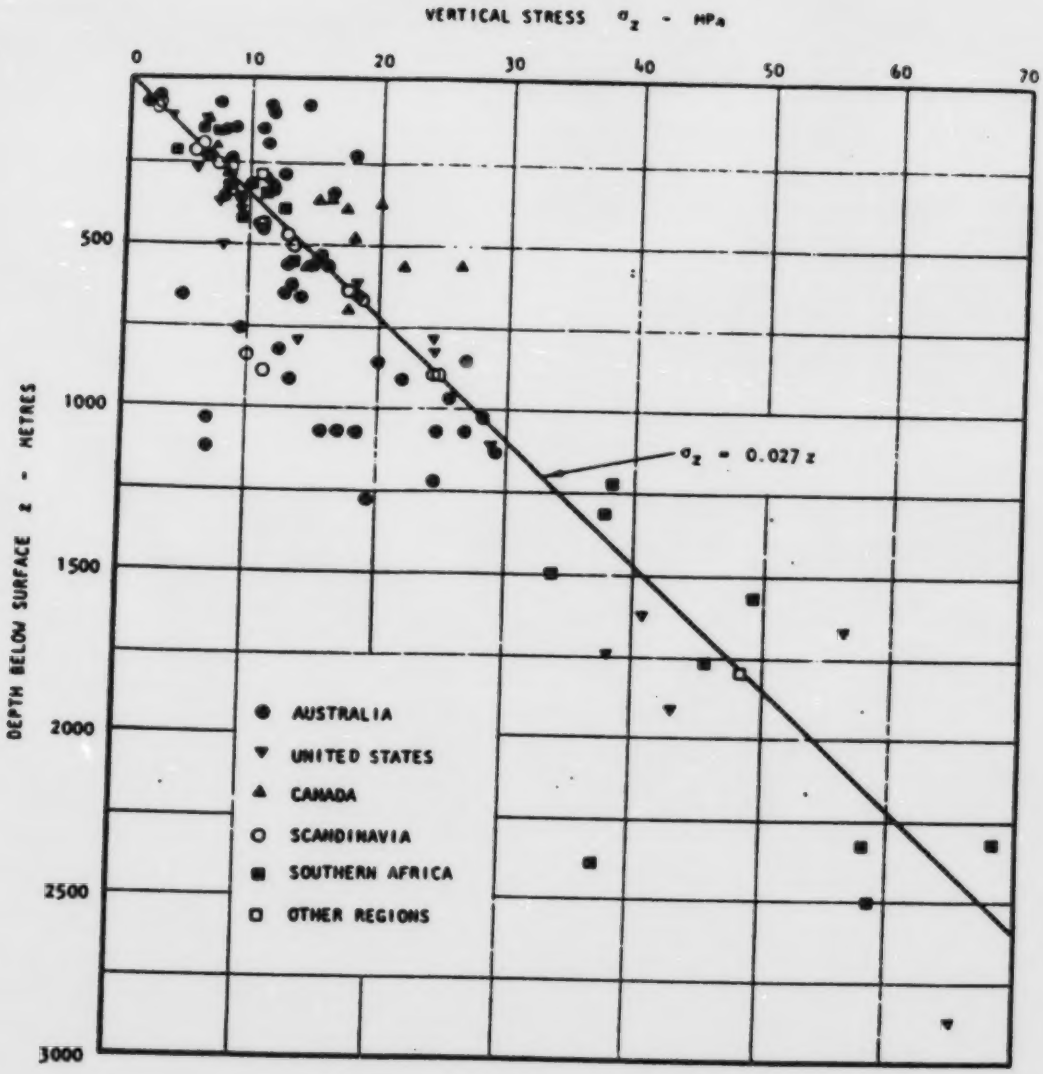
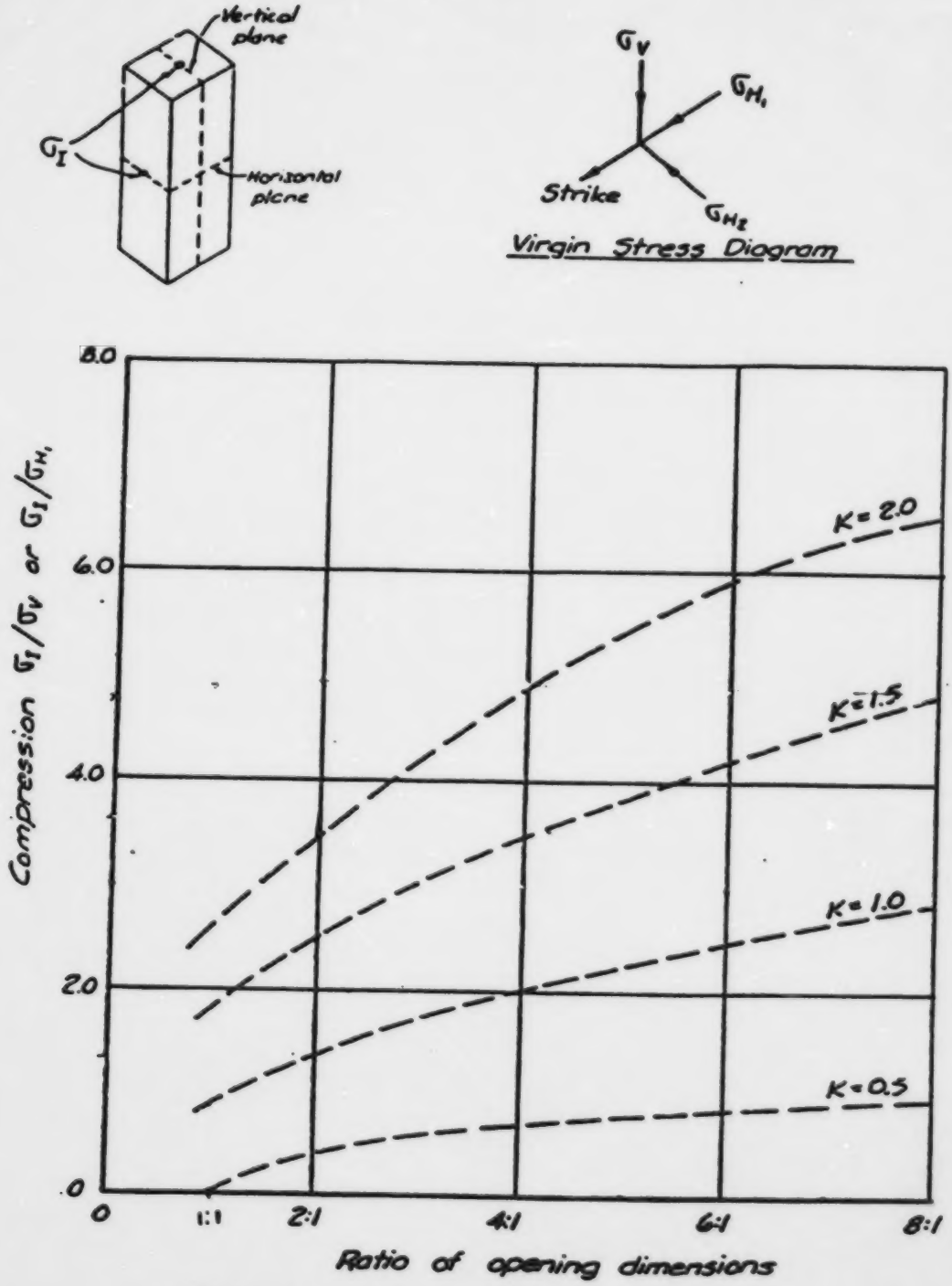


Figure 4.1

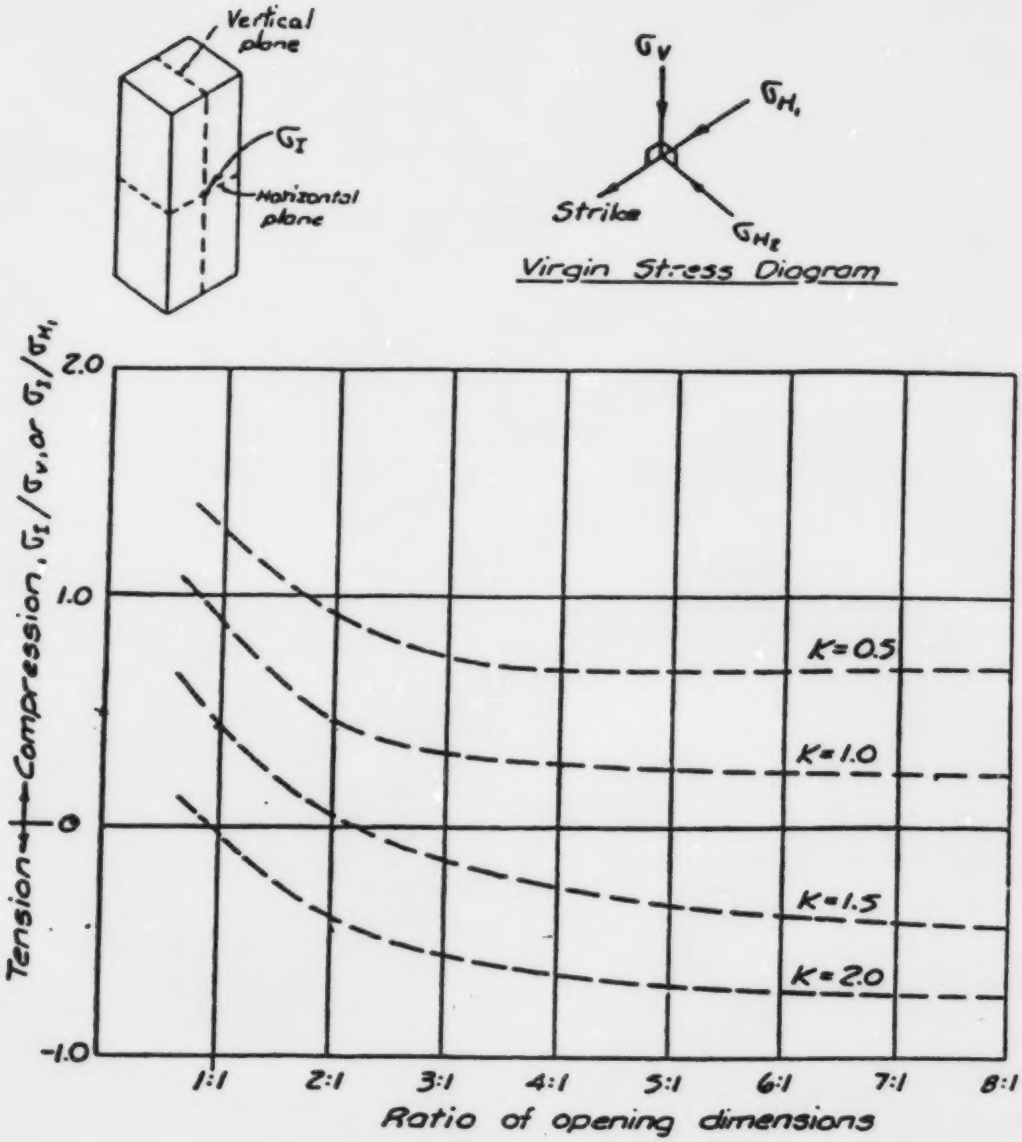
Mathews et Al[8]



LEGEND

σ_I = Induced stress
σ_V = Vertical virgin stress
σ_{H_1} = Horizontal virgin stress on strike
σ_{H_2} = Horizontal virgin stress normal to strike
Horizontal plane, $K = \sigma_{H_2}/\sigma_{H_1}$
Vertical plane, $K = \sigma_{H_2}/\sigma_V$

Figure 4.2
Mathews et Al[8]



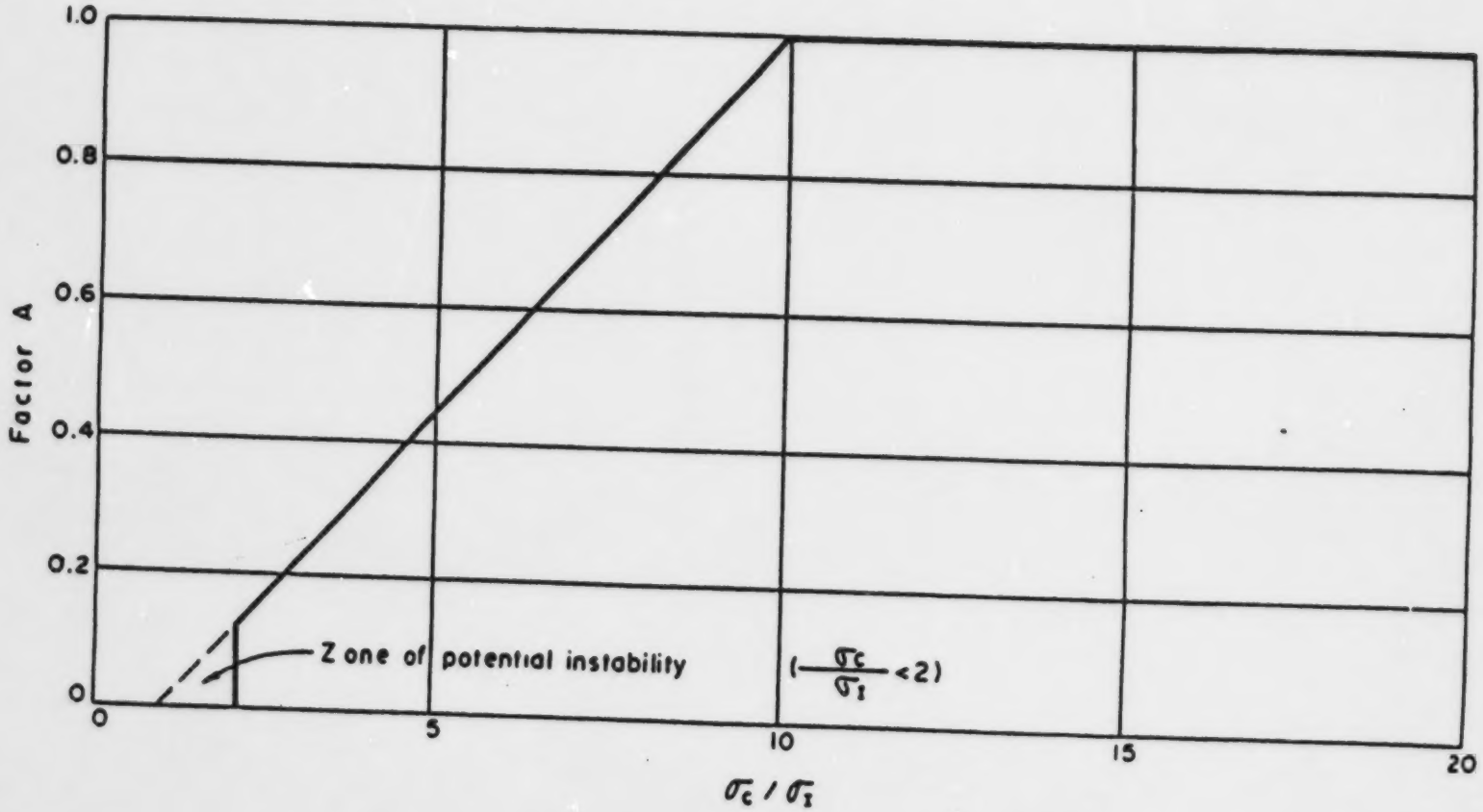
LEGEND

σ_I = Induced stress
σ_V = Vertical virgin stress
σ_{H_1} = Horizontal virgin stress on strike
σ_{H_2} = Horizontal virgin stress normal to strike
Horizontal plane, $K = \sigma_{H_2}/\sigma_{H_1}$
Vertical plane, $K = \sigma_{H_2}/\sigma_V$

Figure 4.3

Mathews et Al[8]

FIGURE 4.4 Rock Stress Factor A. (after Mathews et Al. 1981)



σ_c • Uniaxial compressive strength of intact rock
σ_I • Induced compressive stress

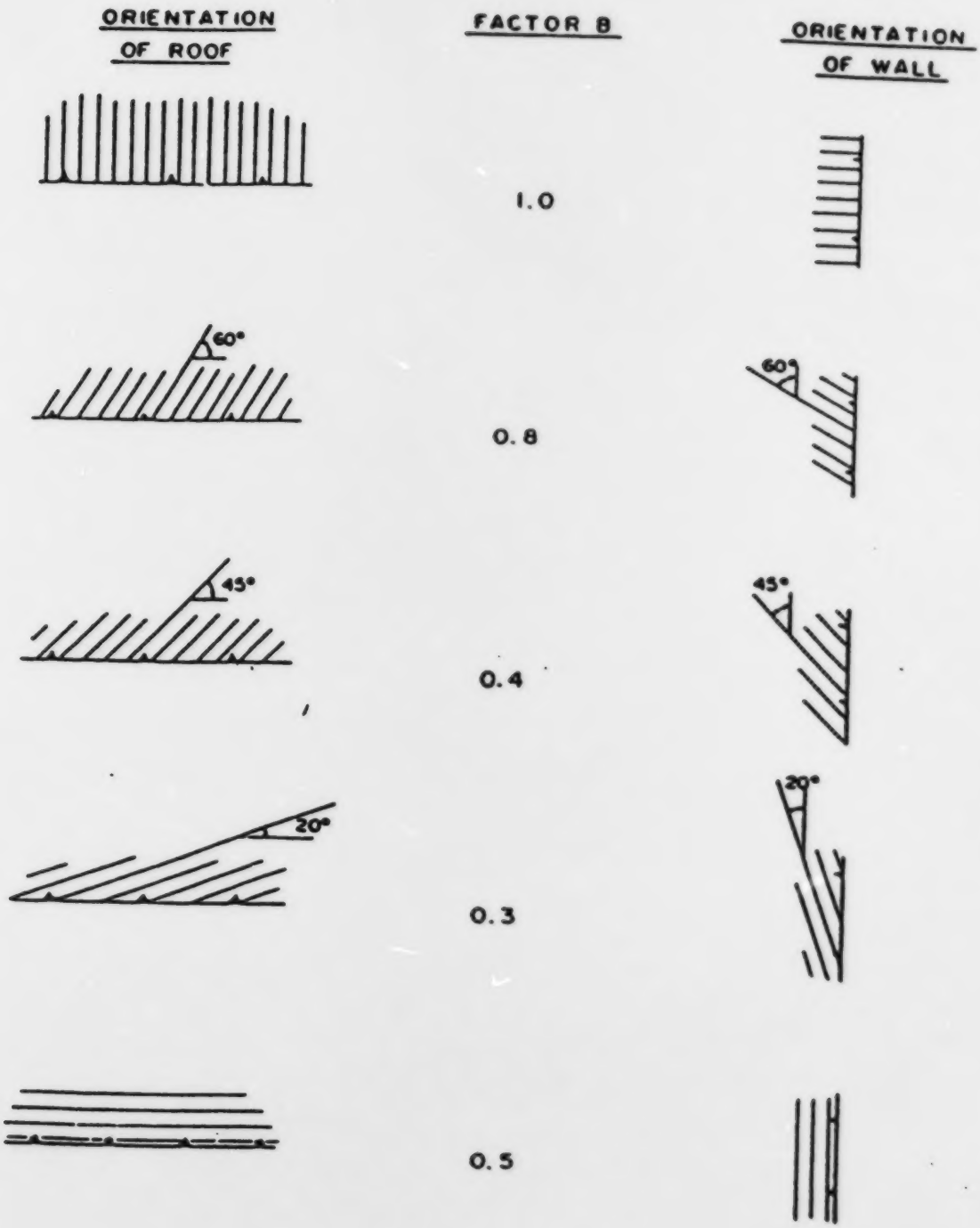
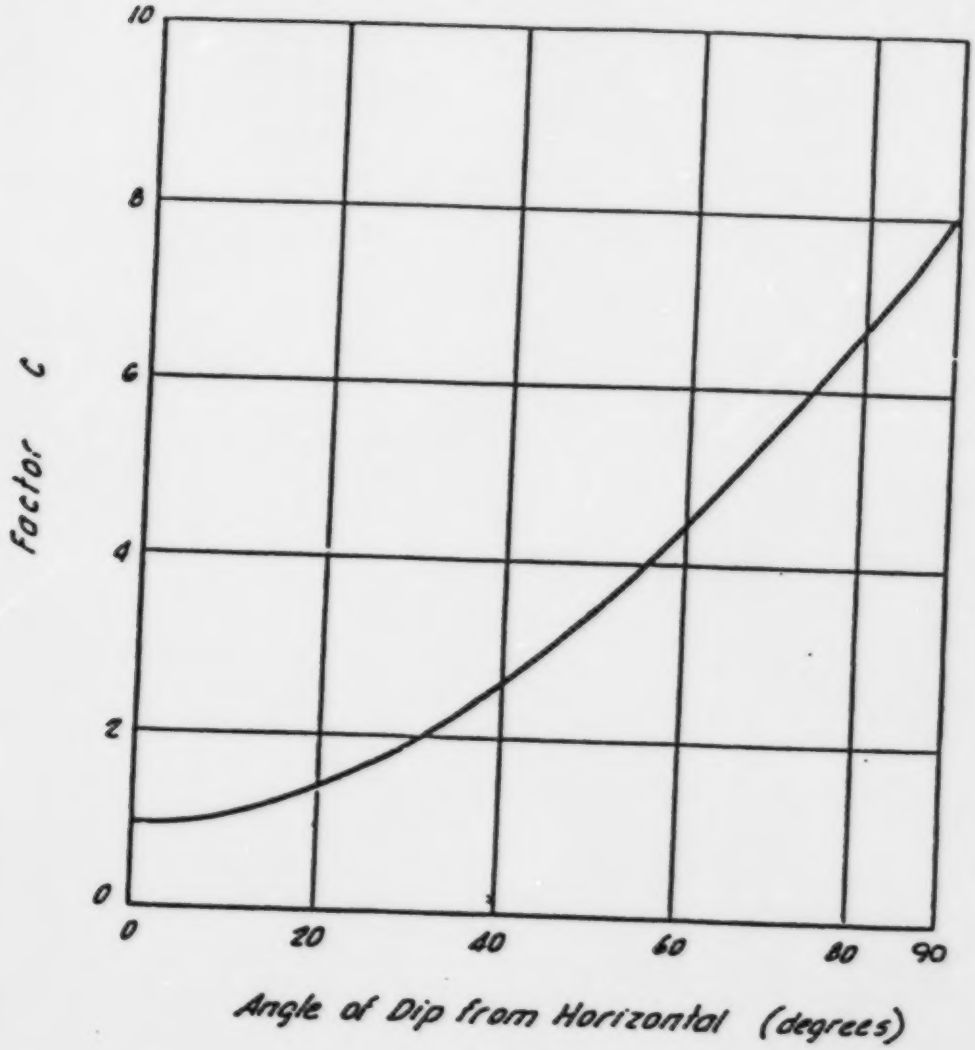


FIGURE 4.5 ROCK DEFECT ORIENTATION FACTOR (B)

(after Mathews et Al. 1981)

FIGURE 4.6 Design Surface Orientation Factor (C)
(after Mathews et Al. 1981)



Factor C = 8 - 7 Cosine (angle of dip)

Trow

Pamour Mine

Rock Stress Factor (A)

Dip of the orebody is dipping vertically.

Depth to centre of stope from surface is approximately 37.5 m.

Vertical Stress estimated at $\sigma v = 0.027 \times 37.5$
$= 1.01$ MPa

$K = \frac{\sigma H}{\sigma v}$ $K = 1.5$

Horizontal Stress $\sigma H = 1.5 \times 1.01 = 1.52$ MPa

First step is to calculate induced stresses in the back (vertical plane) and the strike end (horizontal plane).

Vertical Plane

H:S is 4.1:1

$\sigma I:\sigma v$ is estimated at 3.6 from figure 4-2.

$\frac{\sigma I}{\sigma v} = 3.6$ $\sigma I = 3.6 \times 1.01$
$= 3.64$ MPa

Uniaxial Compressive Strength of the rock $\sigma c = 143$ MPa

$\frac{\sigma c}{\sigma I} = \frac{143}{3.64} = 39.35$ A = 1.0 from figure 4-4

Horizontal Plane

L:S is 2:1

$\sigma H1 = 1.52$ MPa

$\sigma H2 = 1.52$ MPa

$k = \frac{\sigma H2}{\sigma H1} = 1.0$ $\frac{\sigma I}{\sigma H} = 2.0$ from figure 4-2

$\sigma I = 2.0 \times 1.52 = 3.04$ MPa

$\frac{\sigma c}{\sigma I} = \frac{143}{3.04} = 47.04$ A = 1.0 from figure 4-4

Next step is to calculate the induced stresses in the hangingwall and footwall considering the vertical and horizontal mid-stope planes. The lowest value of the estimated rock stress factor (A) is used.

Vertical Plane

H:S is 4.1:1

$\sigma I : \sigma v$ is estimated at -0.2 from figure 4-3

As the value is negative it is set to zero and $\sigma I = 0$ therefore referring to figure 4-4. A = 1.0. It should be noted however that horizontal joints intersecting the hangingwall will open as the induced stresses at the center of the hangingwall span is tensile.

Horizontal Plane

L:S = 2:1 K = 1.0

$\frac{\sigma I}{\sigma H} = 0.5$ from figure 4-3

$\sigma I = 0.5 \times 1.52 = 0.76$ MPa

$\frac{\sigma c}{\sigma} = \frac{143}{0.76} = 188.16$ MPa A = 1.0 from figure 4-4

Summarizing the rock stress factors (A) to be used are listed below.

Back	-A = 1.0
Vertical End	-A = 1.0
H/W	-A = 1.0
F/W	-A = 1.0

Rock Defect Orientation Factor (B)

The principal joint set is dipping at 90° and the orebody is 9.2 m wide and dips at 90°. Referring to figure 4-5, the orientation factor for the exposed surfaces are as follows:

Exposed Surface	Value of (B)
Back	1
Vertical End	1
H/W	0.5
F/W	0.5

Design Surface Orientation Factor (C)

The hangingwall and footwall dip at 90° hence the design surface orientation factors for the exposed surfaces are as follows: figure 4-6.

Exposed Surface	Value of (C)
Back	1.0
Vertical End	8.0
H/W	6.8
F/W	8.0

Stability Nunber (N) = Q' x A x B x C

Back N = 12.8 x 1.0 x 1.0 x 1.0
= 12.8

Shape Factor (S)

Back

$$S = \frac{\text{Area of Surface}}{\text{Perimeter of Surface}}$$

$$= \frac{9.2 \times 18.4}{(9.2 \times 2) + (18.4 \times 2)}$$

$$= 3.1 \text{ m}$$

Summary of Factors For Pamour Mine

Exposed Surface	A	B	C	Q'	N	S
Back	1.0	1.0	1.0	12.8	12.8	3.1
Vertical End	1.0	1.0	8.0	12.8	102.4	3.7
F/W	1.0	0.5	8.0	2.2	8.8	6.2
H/W	1.0	0.5	6.8	5.3	18.0	6.2

Trow

Bousquet Mine

Rock Stress Factor (A)

Dip of the orebody is approximately 75°

Depth to centre of stope from surface is approximately 43 m.

Vertical Stress estimated at $\sigma v = 0.027 \times 43$
$= 1.16$ MPa

$K = \frac{\sigma H}{\sigma v}$ $K = 1.5$

Horizontal Stress $\sigma H = 1.5 \times 1.16 = 1.74$ MPa

First step is to calculate induced stresses in the back (vertical plane) and the strike end (horizontal plane).

Vertical Plane

H:S is 8.1

$\sigma I:\sigma v$ is estimated at 5.0 from figure 4-2

$\frac{\sigma I}{\sigma v} = 5.0$ $\sigma I = 5.0 \times 1.16$
$= 5.8$ MPa

Uniaxial Compressive Strength of the rock $\sigma c = 53.78$ MPa

$\frac{\sigma c}{\sigma I} = \frac{53.78}{5.8} = 9.3$ A = 0.96 from figure 4-4

Horizontal Plane

L:S is 2:1

$\sigma H1 = 1.74$ MPa

$\sigma H2 = 1.74$ MPa

$K = \frac{\sigma H2}{\sigma H1} = 1.0$ $\frac{\sigma I}{\sigma H} = 1.4$ from figure 4-2

$\sigma I = 1.4 \times 1.74 = 2.44$ MPa

$\frac{\sigma c}{\sigma I} = \frac{53.78}{2.44} = 22.1$ A = 1.0 from figure 4-4

Next step is to calculate the induced stresses in the hangingwall and footwall considering the vertical and horizontal mid-stope planes. The lowest value of the estimated rock stress factor (A) is used.

Vertical Plane

H:S is 8:1

$\sigma I:\sigma v$ is estimated at -0.5 from figure 4-3

As the value is negative; it is set to zero and $\sigma I = 0$ therefore referring to figure 4-4. A = 1.0. It should be noted however that horizontal joints intersecting the hangingwall will open as the induced stresses at the center of the hangingwall span is tensile.

Horizontal Plane

L:S = 2:1 K = 1.0

$\frac{\sigma I}{\sigma H} = 0.5$ from figure 4-3

$\sigma I = 0.5 \times 1.74 = 0.87$ MPa

$\frac{\sigma c}{\sigma I} = \frac{53.78}{0.87} = 61.8$ A = 1.0 from figure 4-4

Summarizing the rock stress factors (A) to be used are listed below.

Back	-A = 0.96
Vertical End	-A = 1.0
H/W	-A = 1.0
F/W	-A = 1.0

Rock Defect Orientation Factor (B)

The principal joint set is dipping at 45° and the orebody is 5 m wide and dips at 75°. Referring to Figure 4-5, the orientation factor for the exposed surfaces are as follows:

Exposed Surface	Value of (B)
Back	0.4
Vertical End	0.4
H/W	0.3
F/W	0.3

Design Surface Orientation Factor (C)

The hangingwall and footwall dip at 75° hence the design surface orientation factors for the exposed surfaces are as follows: figure 4-6.

Exposed Surface	Value of (C)
Back	1
Vertical End	8
H/W	5.5
F/W	8

Stability Number (N) = Q' x A x B x C

Back N = 2.9 x 0.96 x 0.4 x 1
= 1.1

Shape Factor (S)

Back

$$S = \frac{\text{Area of Surface}}{\text{Perimeter of Surface}}$$

$$= \frac{10 \times 5}{(2 \times 5) + (2 \times 10)}$$

$$= 1.7$$

Summary of Factors For Bousquet Mine

Exposed Surface	A	B	C	Q'	N	S
Back	0.96	0.4	1.0	2.9	1.1	1.7
Vertical End	1.0	0.4	8.0	2.9	9.3	2.2
F/W	1.0	0.3	8.0	1.5	3.8	4.0
H/W	1.0	0.3	5.5	1.5	2.4	4.0

Trow

Belmoral Mine

Rock Stress Factor (A)

Dip of the orebody is approximately 70°.

Depth to centre of stope from surface is approximately 38.1 m.

Vertical Stress estimated at $\sigma v = 0.027 \times 38.1$
$= 1.03$ MPa

$K = \frac{\sigma H}{\sigma v}$ $K = 1.5$

Horizontal Stress $\sigma H = 1.5 \times 1.03 = 1.55$ MPa

First step is to calculate induced stresses in the back (vertical plane) and the strike end (horizontal plane).

Vertical Plane

H:S is 15.3:1 use 8:1

$\sigma I:\sigma v$ is estimated at 5.0 from figure 4-2

$\frac{\sigma I}{\sigma v} = 5.0$ $\sigma I = 5.0 \times 1.03$
$= 5.15$ MPa

Uniaxial Compression Strength of the rock $\sigma c = 76$ MPa

$\frac{\sigma c}{\sigma I} = \frac{76}{5.15} = 14.89$ MPa A = 1.0 from figure 4-4

Horizontal Plane

L:S is 2:1

$\sigma H1 = 1.55$ MPa

$\sigma H2 = 1.55$ MPa

$K = \frac{\sigma H2}{\sigma H1} = 1.0$ $\frac{\sigma I}{\sigma} = 1.4$ from figure 4-2

$\sigma I = 1.4 \times 1.55 = 2.17$ MPa

$\frac{\sigma c}{\sigma I} = \frac{76}{2.17} = 35$ A = 1.0 from figure 4-4

Next step is to calculate the induced stresses in the hangingwall and footwall considering the vertical and horizontal mid-stope planes. The lowest value of the estimated rock stress factor (A) is used.

Vertical Plane

H:S is 15.3:1 K = 1.5

$\sigma I:\sigma v$ is estimated at -0.5 from figure 4-3

As the value is negative, it is set to zero and $\sigma I = 0$ therefore referring to figure 4-4. A = 1.0. It should be noted however that horizontal joints intersecting the hangingwall will open as the induced stresses at the center of the hangingwall span is tensile.

Horizontal Plane

L:S = 2:1 K = 1.0

$\frac{\sigma I}{\sigma H} = 0.5$ from figure 4-3

$\sigma I = 0.5 \times 1.55 = 0.78$ MPa

$\frac{\sigma c}{\sigma} = \frac{76}{0.78} = 98.1$ MPa A = 1.0 from figure 4-4

Summarizing the rock stress factors (A) to be used are listed below.

Back	-A = 1.0
Vertical End	-A = 1.0
H/W	-A = 1.0
F/W	-A = 1.0

Rock Defect Orientation Factor (B)

The principal joint set is dipping at 60° and the orebody is 3.0 m wide and dips at 70°. Referring to figure 4-5, the orientation factor for the exposed surfaces are as follows:

Exposed Surface	Value of (B)
Back	0.8
Vertical End	0.8
H/W	0.3
F/W	0.3

Design Surface Orientation Factor (C)

The hangingwall and footwall dip at 70° hence the design surface orientation factors for the exposed surfaces are as follows: figure 4-6.

Exposed Surface	Value of (C)
Back	1
Vertical End	8
H/W	5.8
F/W	8.0

Stability Number (N) = Q' x A x B x C

Back

$$N = 1.2 \times 1.0 \times 0.8 \times 1.0$$
$$= 0.96$$

Shape Factor (S)

Back

$$S = \frac{\text{Area of Surface}}{\text{Perimeter of Surface}}$$
$$= \frac{6 \times 3}{(6 \times 2) + (3 \times 2)}$$
$$= 1.0$$

Summary of Factors For Belmoral Mine

Exposed Surface	A	B	C	Q'	N	S
Back	1.0	0.8	1	1.2	0.96	1.0
Vertical End	1.0	0.8	8	1.5	9.6	1.4
F/W	1.0	0.3	8	2.5	6	2.6
H/W	1.0	0.3	5.8	2.5	4.4	2.6

Trow

Chimo Mine

Rock Stress Factor (A)

Dip of the orebody is approximately 70°.

Depth to centre of stope from surface is approximately 90 m.

Vertical Stress estimated at $\sigma v = 0.027 \times 90$
$= 2.43$ MPa

$K = \frac{\sigma H}{\sigma v}$ $K = 1.5$

Horizontal Stress $\sigma H = 1.5 \times 2.43 = 3.65$ MPa

First step is to calculate induced stresses in the back (vertical plane) and the strike end (horizontal plane).

Vertical Plane

H:S is 10:1 use 8:1
$\sigma I:\sigma v$ is estimated at 5.0 from figure 4-2
$\frac{\sigma I}{\sigma v} = 5.0$ $\sigma I = 5.0 \times 2.43$
$= 12.15$ MPa

Uniaxial Compressive Strength of the rock $\sigma c = 39.79$ MPa

$\frac{\sigma c}{\sigma I} = \frac{39.79}{12.15} = 3.27$ A = 0.20 from figure 4-4

Horizontal Plane

L:S is 2:1

$\sigma H1 = 3.65$ MPa

$\sigma H2 = 3.65$ MPa

$K = \frac{\sigma H2}{\sigma H1} = 1.0$ $\quad \frac{\sigma I}{\sigma H} = 1.4$ from figure 4-2

$\sigma I = 1.4 \times 3.65 = 5.10$ MPa

$\frac{\sigma c}{\sigma I} = \frac{39.79}{5.10} = 7.80$ $\quad A = 0.7$ from figure 4-4

Next step is to calculate the induced stresses in the hangingwall and footwall considering the vertical and horizontal mid-stope planes. The lowest value of the estimated rock stress factor (A) is used.

Vertical Plane

H:S is 10:1

$\sigma I:\sigma v$ is estimated at -0.5 from figure 4-3

As the value is negative; it is set to zero and $\sigma I = 0$ therefore referring to figure 4-4. A = 1.0. It should be noted however that horizontal joints intersecting the hangingwall will open as the induced stresses at the center of the hangingwall span is tensile.

Horizontal Plane

L:S = 2:1 K = 1.0

$\frac{\sigma I}{\sigma H} = 0.5$ from figure 4-3

$\sigma I = 0.5 \times 3.65 = 1.83$ MPa

$\frac{\sigma c}{\sigma} = \frac{39.79}{1.83} = 21$ MPa $\quad A = 1.0$ from figure 4-4

Summarizing the rock stress factors (A) to be used are listed below.

Back	-A = 0.2
Vertical End	-A = 0.70
H/W	-A = 1.0
F/W	-A = 1.0

Rock Defect Orientation Factor (B)

The principal joint set is dipping at 70° and the orebody is 4.0 m wide and dips at 70°. Referring to figure 4-5, the orientation factor for the exposed surfaces are as follows:

Exposed Surface	Value of (B)
Back	0.8
Vertical End	0.8
H/W	0.5
F/W	0.5

Design Surface Orientation Factor (C)

The hangingwall and footwall dip at 70° hence the design surface orientation factors for the exposed surfaces are as follows: figure 4-6.

Exposed Surface	Value of (C)
Back	1.0
Vertical End	8.0
H/W	5.5
F/W	8.0

Stability Number (N) = Q' x A x B x C

Back N = 1.9 x 0.20 x 0.8 x 1.0
= 0.30

Shape Factor (S)

Back

$$S = \frac{\text{Area of Surface}}{\text{Perimeter of Surface}}$$

$$= \frac{4 \times 8}{(2 \times 4) + (2 \times 8)}$$

$$= 1.3$$

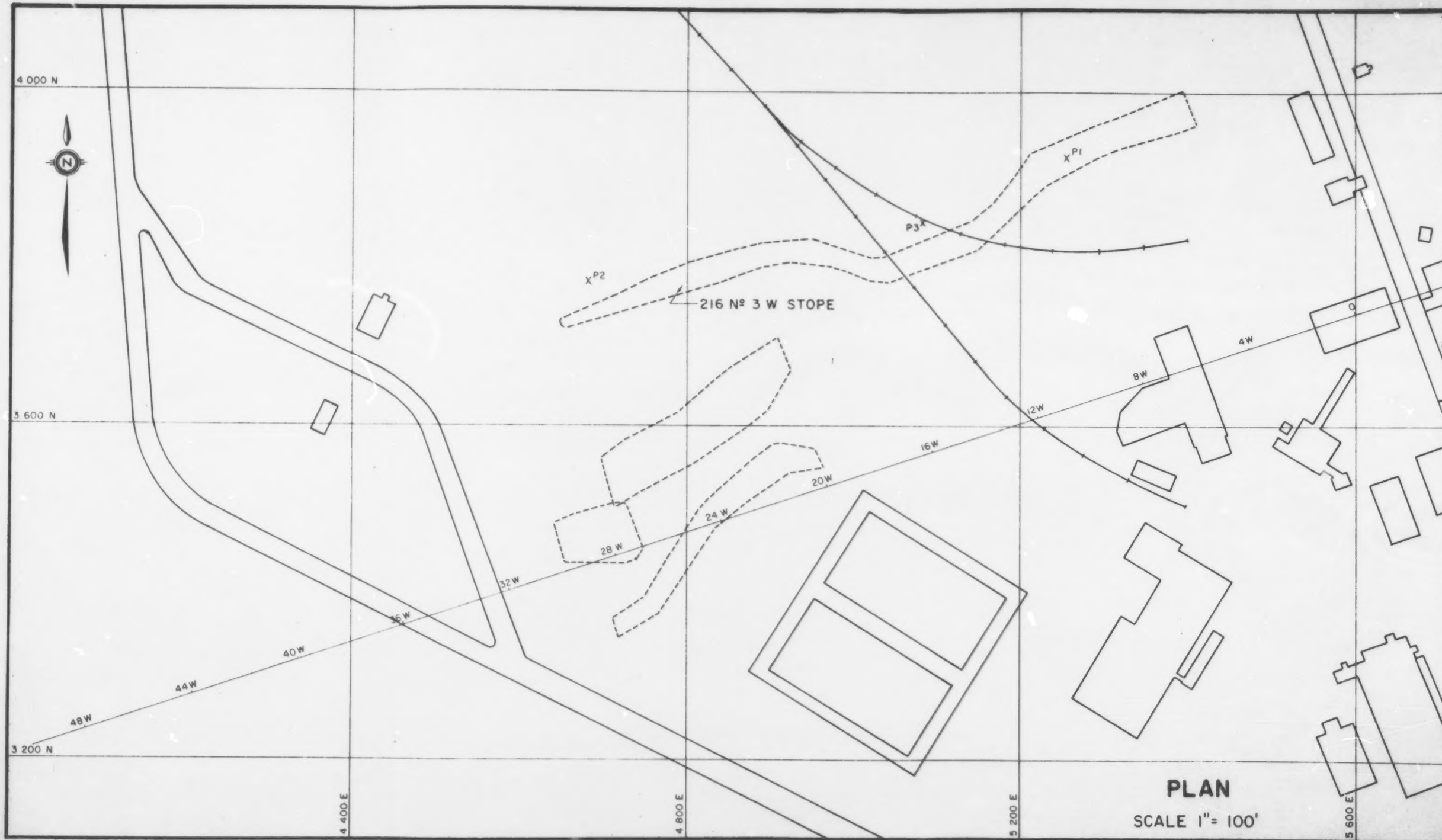
Summary of Factors For Chimo Mine

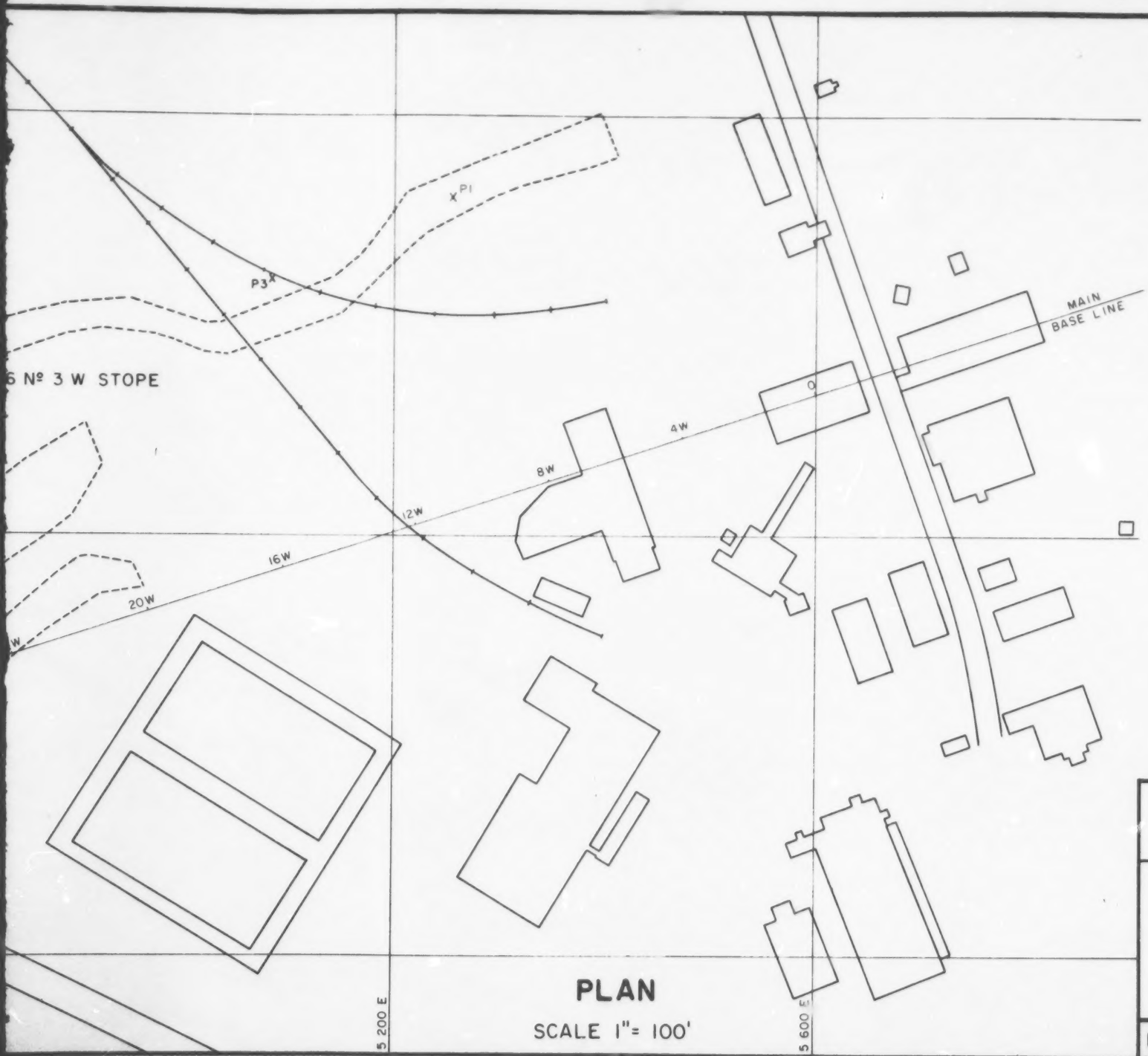
Exposed Surface	A	B	C	Q'	N	S
Back	0.2	0.8	1.0	1.9	0.30	1.3
Vertical End	0.70	0.8	8.0	1.9	8.5	1.8
F/W	0.5	0.5	8.0	4.6	18.4	3.3
H/W	0.5	0.5	5.5	8.1	22.3	3.3

Trow

APPENDIX 5

Mine Cross Sections


4 000 N
3 600 N
3 200 N
4 400 E
4 800 E
5 200 E
5 600 E
N
X P1
X P2
P3
216 Nº 3 W STOPE
0
4W
8W
12W
16W
20W
24 W
28 W
32W
36 W
40W
44W
48W
PLAN
SCALE 1"= 100'



LEGEND

X PROPOSED TEST HOLE LOCATION

NOTES 1) Plan traced from drawing supplied by Giant Yellowknife Mines Ltd.

PLAN

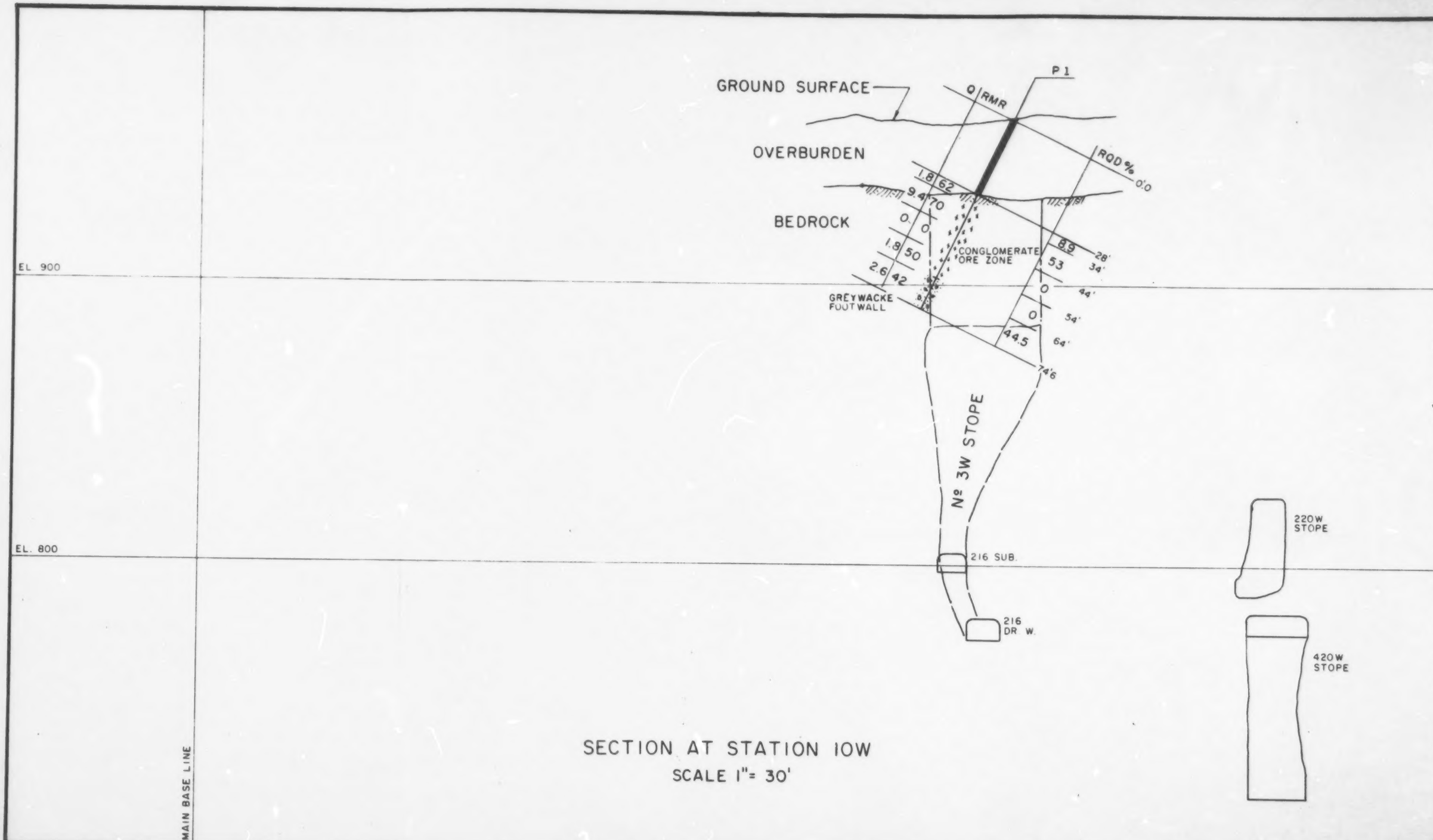
SCALE 1"= 100'

Trow

D.S.S. CROWN PILLAR INVESTIGATION
PAMOUR Nº 1 MINE
TIMMINS, ONTARIO.

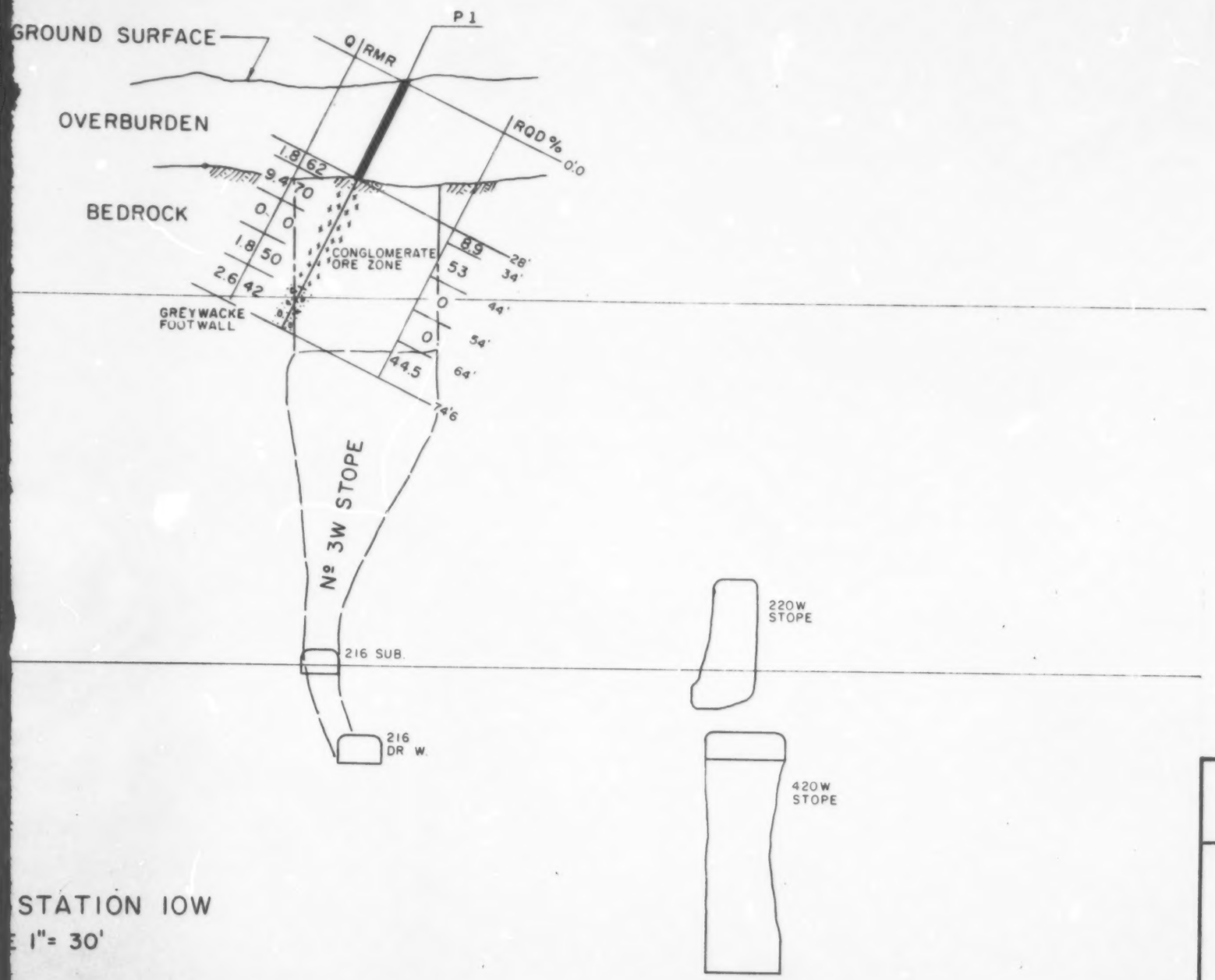
PROJ. S02095R	**DATE** NOVEMBER 1988	**DWG. No.** 1

SEPTEMBER 1989



SECTION AT STATION 10W

SCALE 1"= 30'



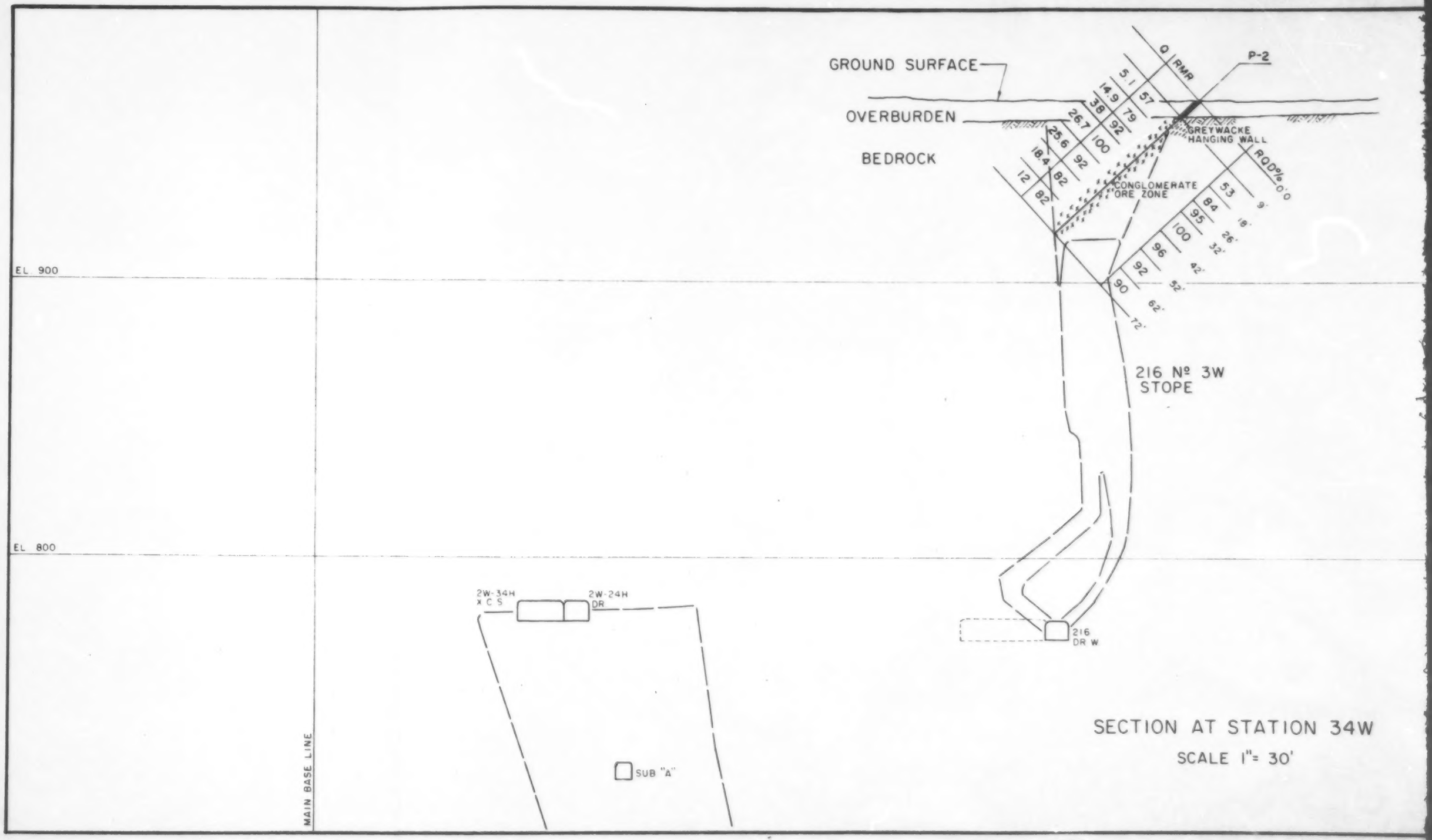
STATION 10W
E 1"= 30'

Trow

D.S.S. CROWN PILLAR INVESTIGATION
PAMOUR Nº 1 MINE
TIMMINS, ONTARIO.

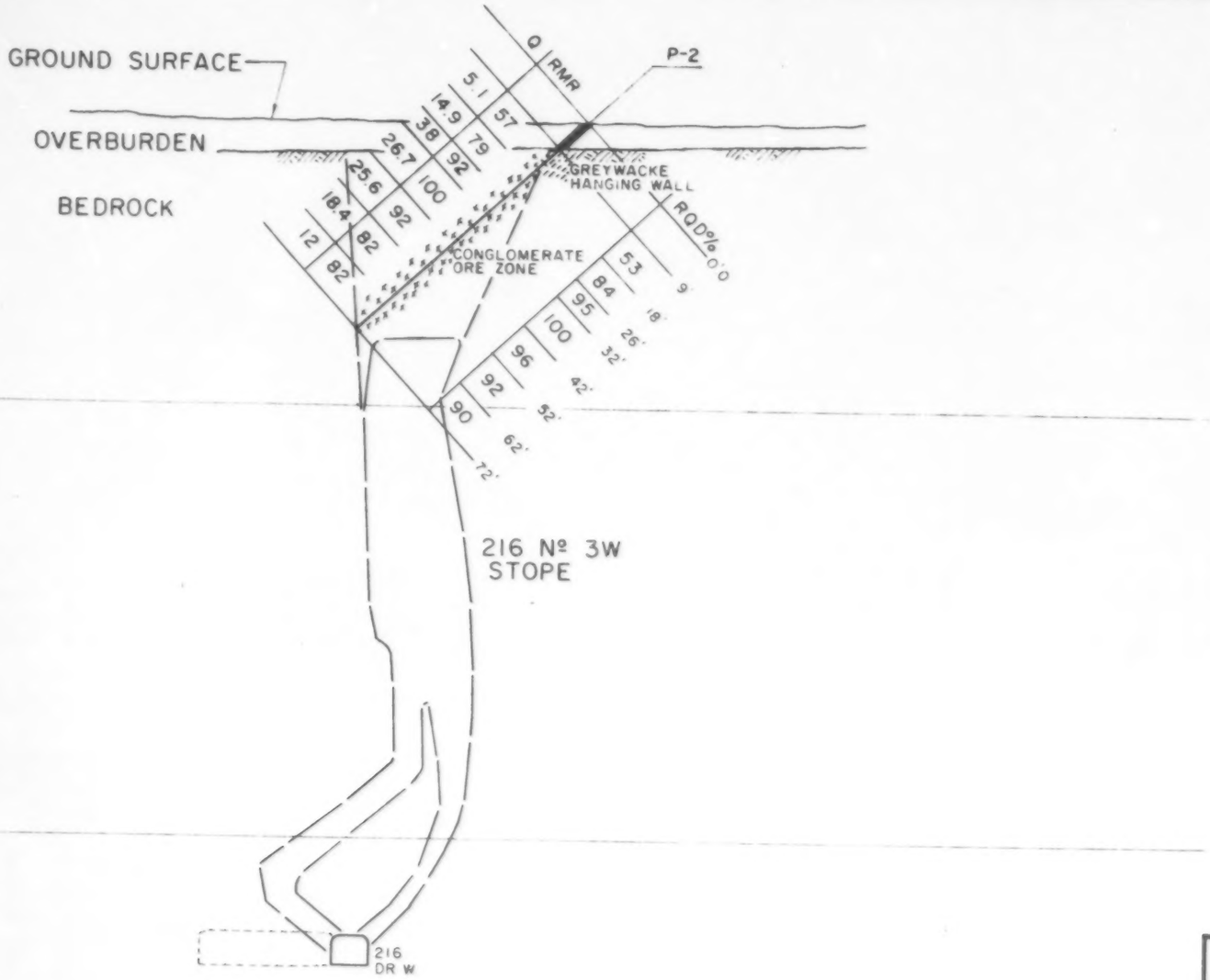
PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 2

SEPTEMBER 1989



SECTION AT STATION 34W

SCALE 1"= 30'



SECTION AT STATION 34W

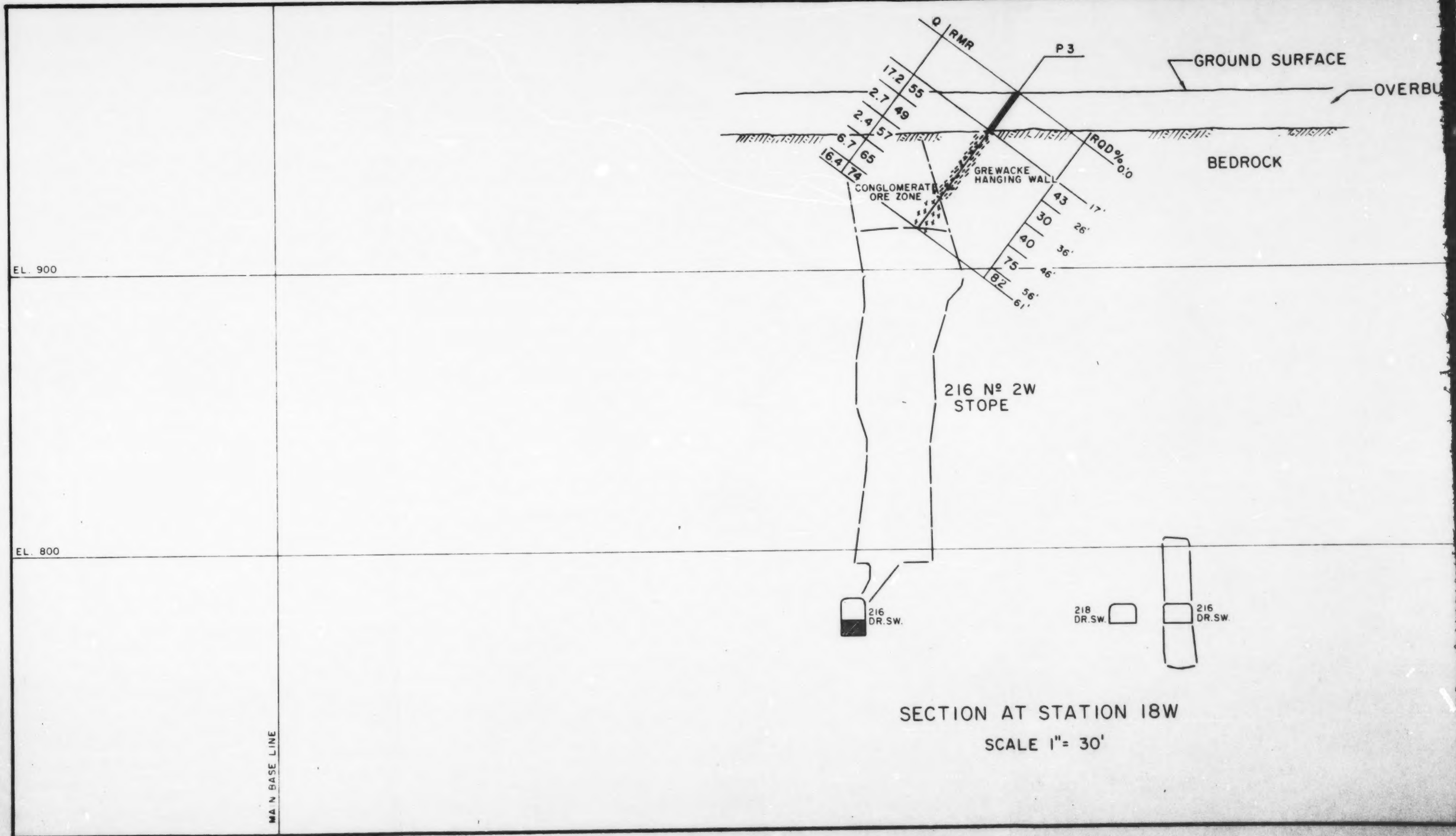
SCALE 1"= 30'

Trow

D.S.S. CROWN PILLAR INVESTIGATION
PAMOUR Nº 1 MINE
TIMMINS, ONTARIO.

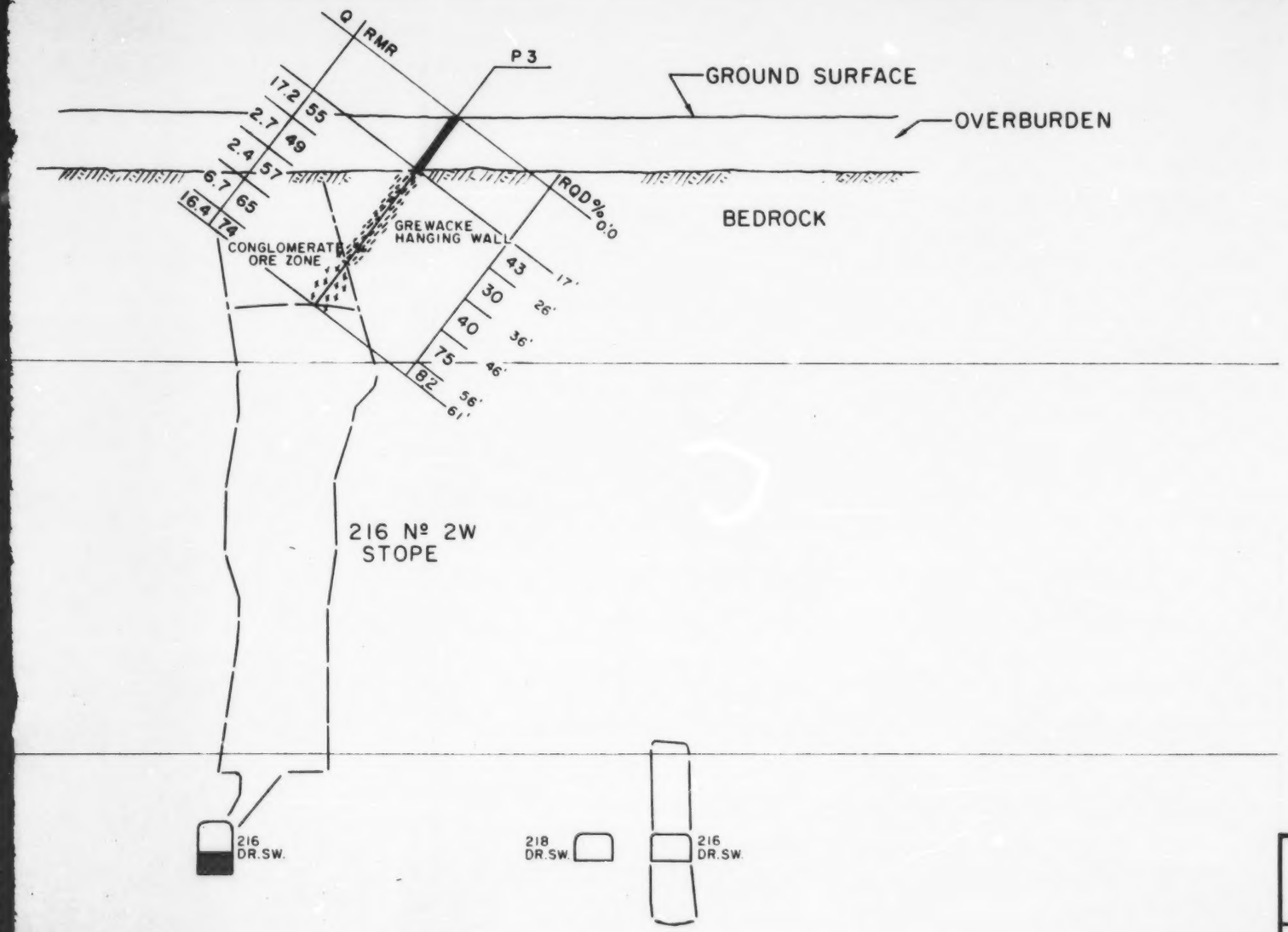
PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 3

SEPTEMBER 1989



SECTION AT STATION 18W

SCALE 1"= 30'



SECTION AT STATION 18W

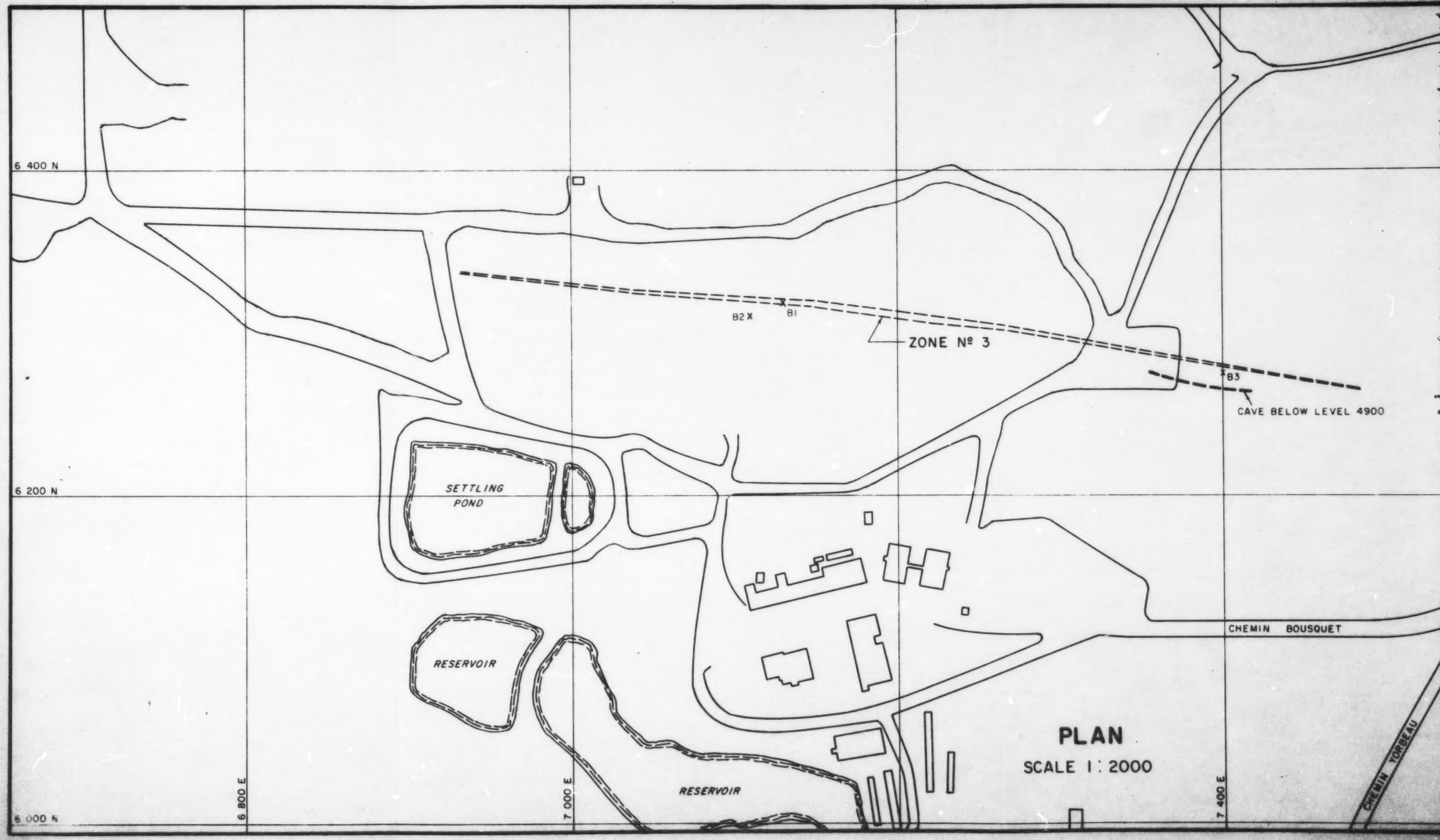
SCALE 1"= 30'

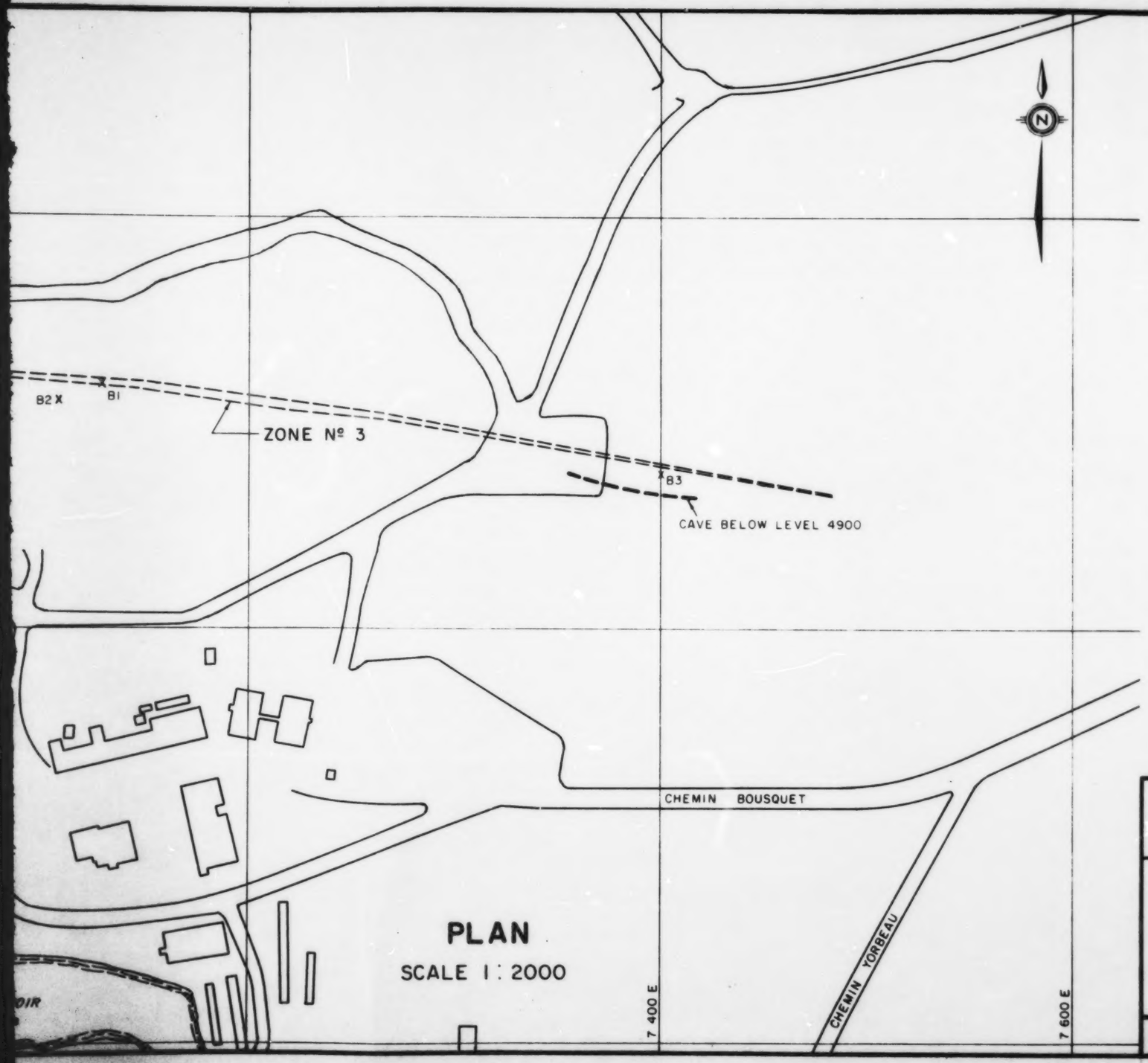
Trow

D.S.S. CROWN PILLAR INVESTIGATION
PAMOUR Nº 1 MINE
TIMMINS, ONTARIO.

PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 4

SEPTEMBER 1989


6 400 N
6 200 N
6 000 N
6 800 E
7 000 E
7 400 E
B2
B1
B3
ZONE Nº 3
CAVE BELOW LEVEL 4900
SETTLING POND
RESERVOIR
RESERVOIR
CHEMIN BOUSQUET
CHEMIN TORBEAU
PLAN
SCALE 1 : 2000



LEGEND

X PROPOSED TEST HOLE LOCATION

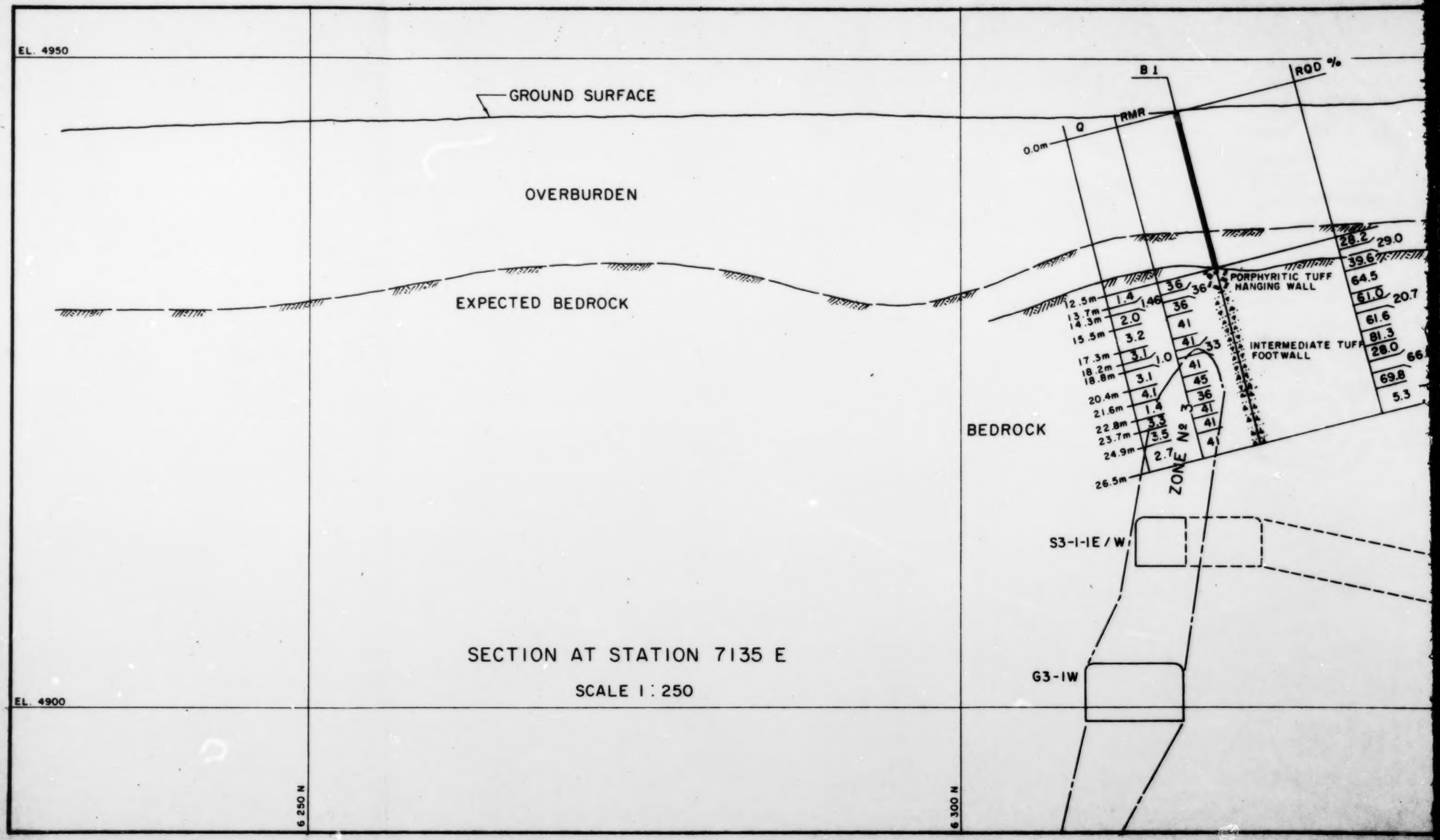
NOTES 1) Plan traced from drawing no. 3-G-58-80P SURFACE supplied by Minerais Loc Ltée, Division Bousquet.

Trow

D.S.S. CROWN PILLAR INVESTIGATION
BOUSQUET MINE
MALARTIC, QUEBEC.

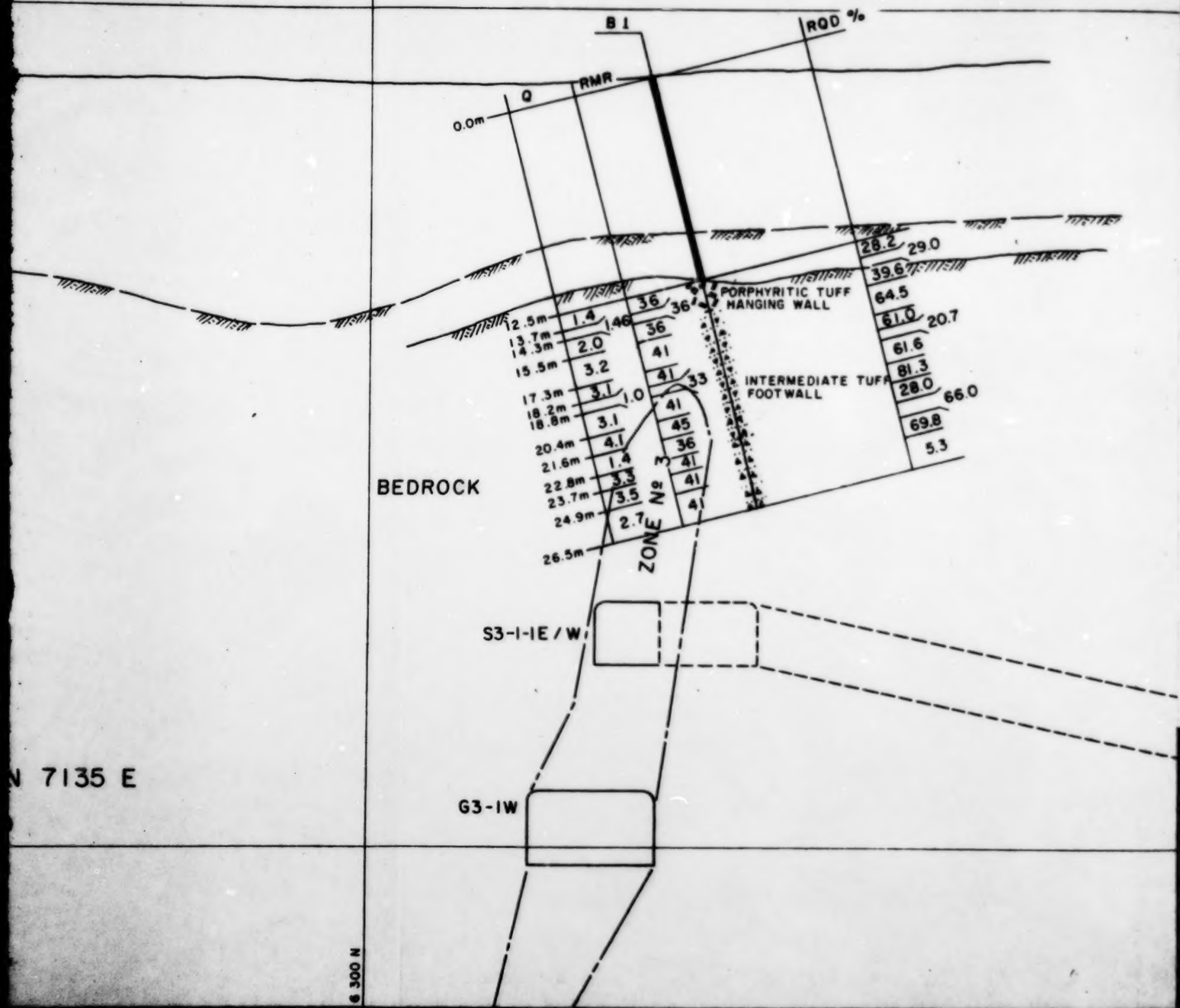
PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 5

SEPTEMBER 1989



SECTION AT STATION 7135 E

SCALE 1 : 250

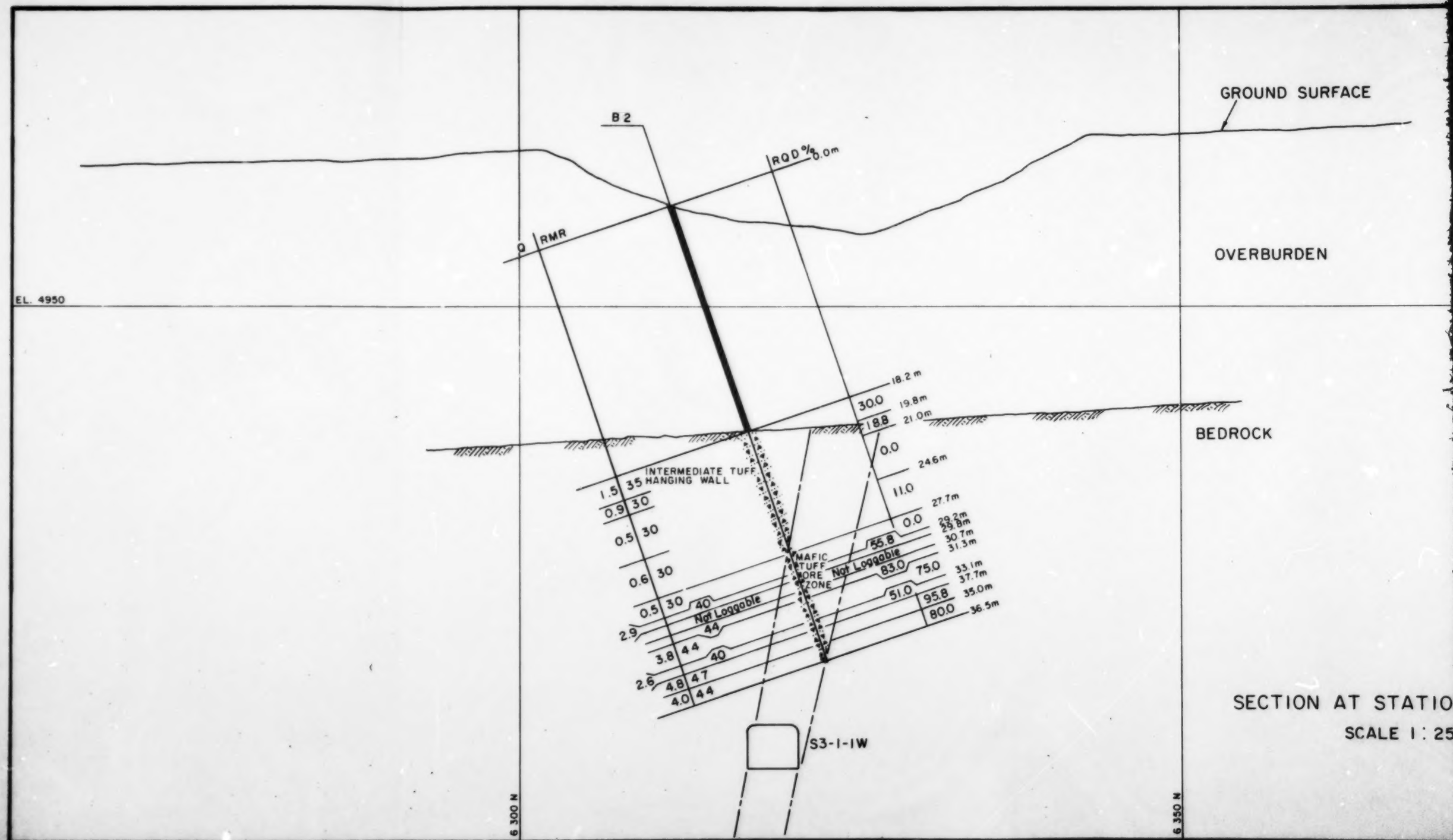


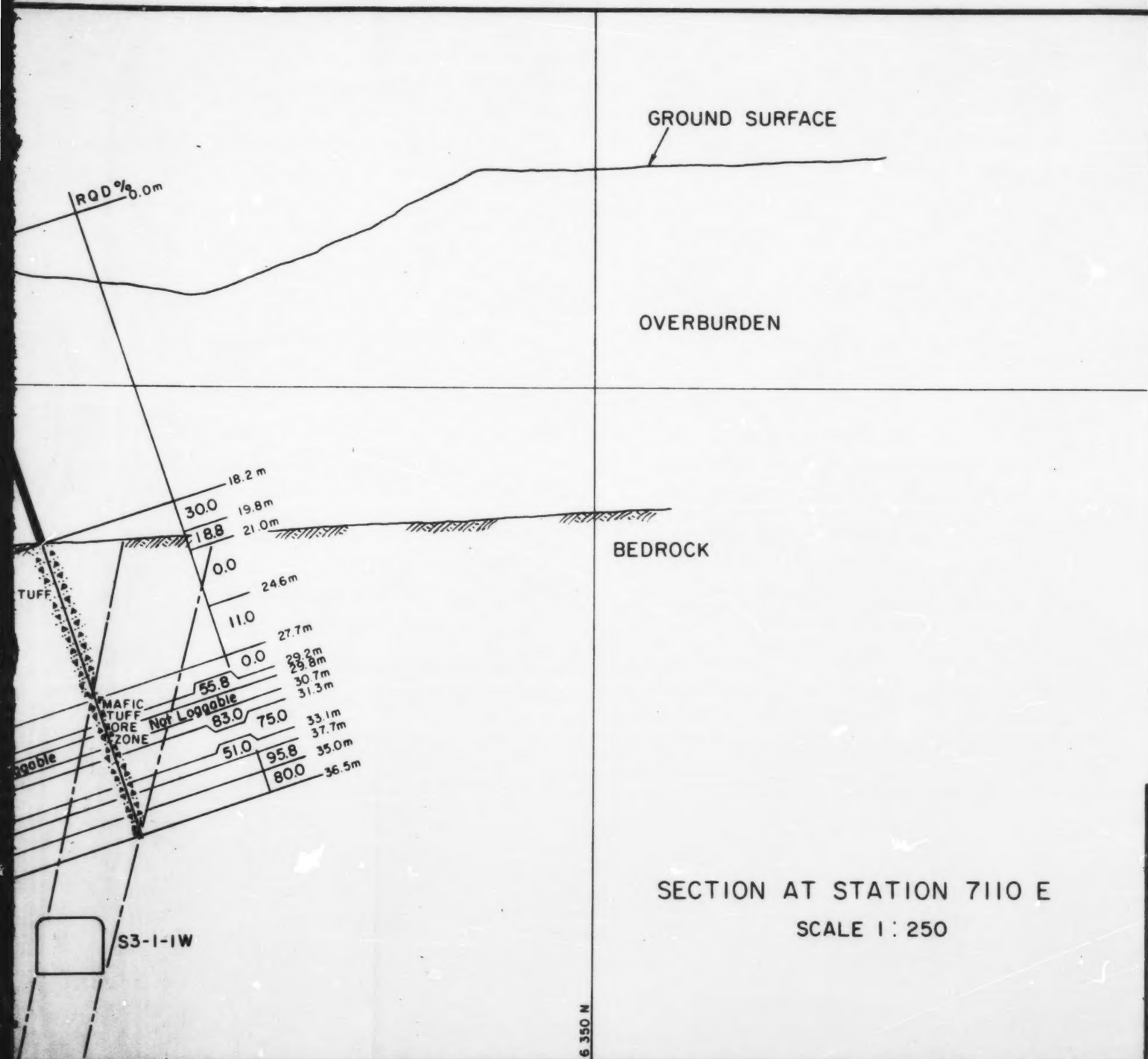
Trow

D.S.S. CROWN PILLAR INVESTIGATION
BOUSQUET MINE
MALARTIC, QUEBEC.

PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 6

SEPTEMBER 1989


GROUND SURFACE
B2
RQD %
6.0m
Q
RMR
OVERBURDEN
EL. 4950
18.2 m
30.0
19.8m
18.8
21.0m
BEDROCK
0.0
24.6m
INTERMEDIATE TUFF
HANGING WALL
1.5
35
11.0
27.7m
0.9
30
0.0
29.2m
29.8m
30.7m
31.3m
0.5
30
55.8
MAFIC
TUFF
ORE
ZONE
Not Loggable
83.0
75.0
33.1m
37.7m
0.6
30
51.0
95.8
35.0m
0.5
30
40
Not Loggable
80.0
36.5m
2.9
44
3.8
44
40
2.6
4.8
47
4.0
44
S3-1-1W
6 300 N
6 350 N
SECTION AT STATIO
SCALE 1 : 25



SECTION AT STATION 7110 E

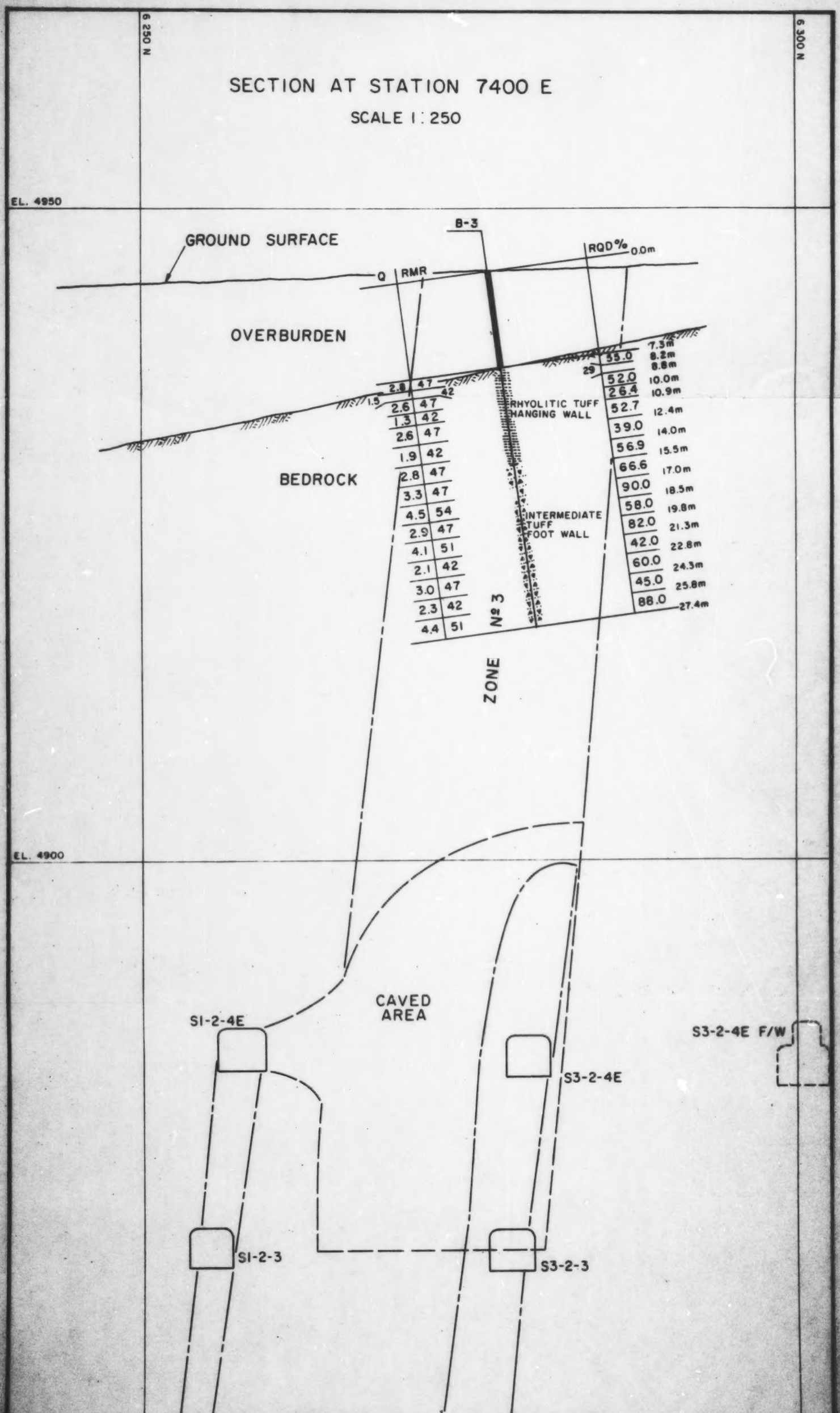
SCALE 1 : 250

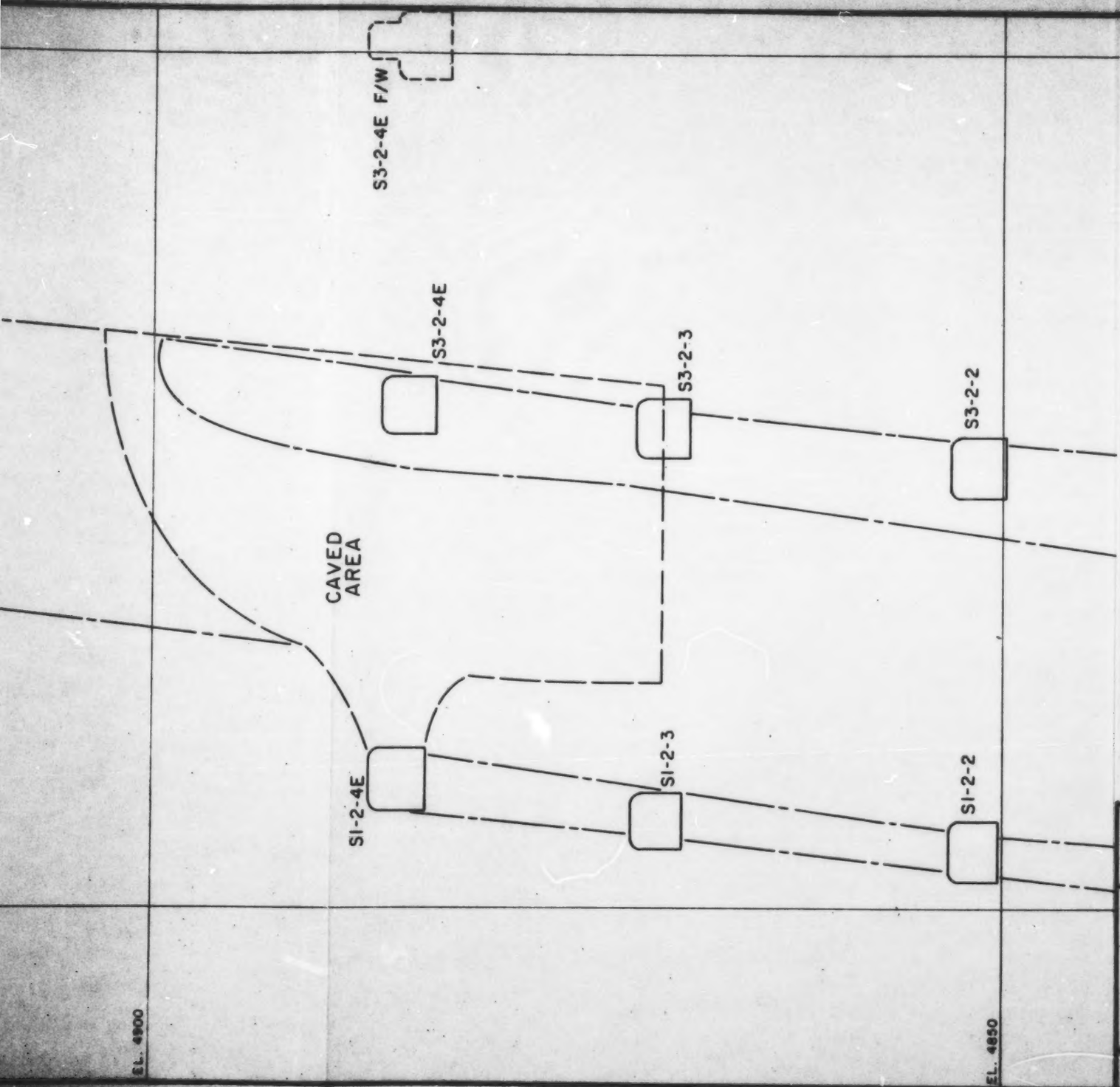
Trow

D.S.S. CROWN PILLAR INVESTIGATION
BOUSQUET MINE
MALARTIC, QUEBEC.

PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 7

SEPTEMBER 1989


SECTION AT STATION 7400 E
SCALE 1:250
6 250 N
6 300 N
EL. 4950
EL. 4900
GROUND SURFACE
B-3
Q
RMR
RQD%
0.0m
OVERBURDEN
BEDROCK
RHYOLITIC TUFF
HANGING WALL
INTERMEDIATE
TUFF
FOOT WALL
ZONE Nº 3
2.8 47
1.5 42
2.6 47
1.3 42
2.6 47
1.9 42
2.8 47
3.3 47
4.5 54
2.9 47
4.1 51
2.1 42
3.0 47
2.3 42
4.4 51
29
55.0 7.3m
8.2m
8.8m
52.0 10.0m
26.4 10.9m
52.7 12.4m
39.0 14.0m
56.9 15.5m
66.6 17.0m
90.0 18.5m
58.0 19.8m
82.0 21.3m
42.0 22.8m
60.0 24.3m
45.0 25.8m
88.0 27.4m
CAVED
AREA
S1-2-4E
S3-2-4E
S3-2-4E F/W
S1-2-3
S3-2-3

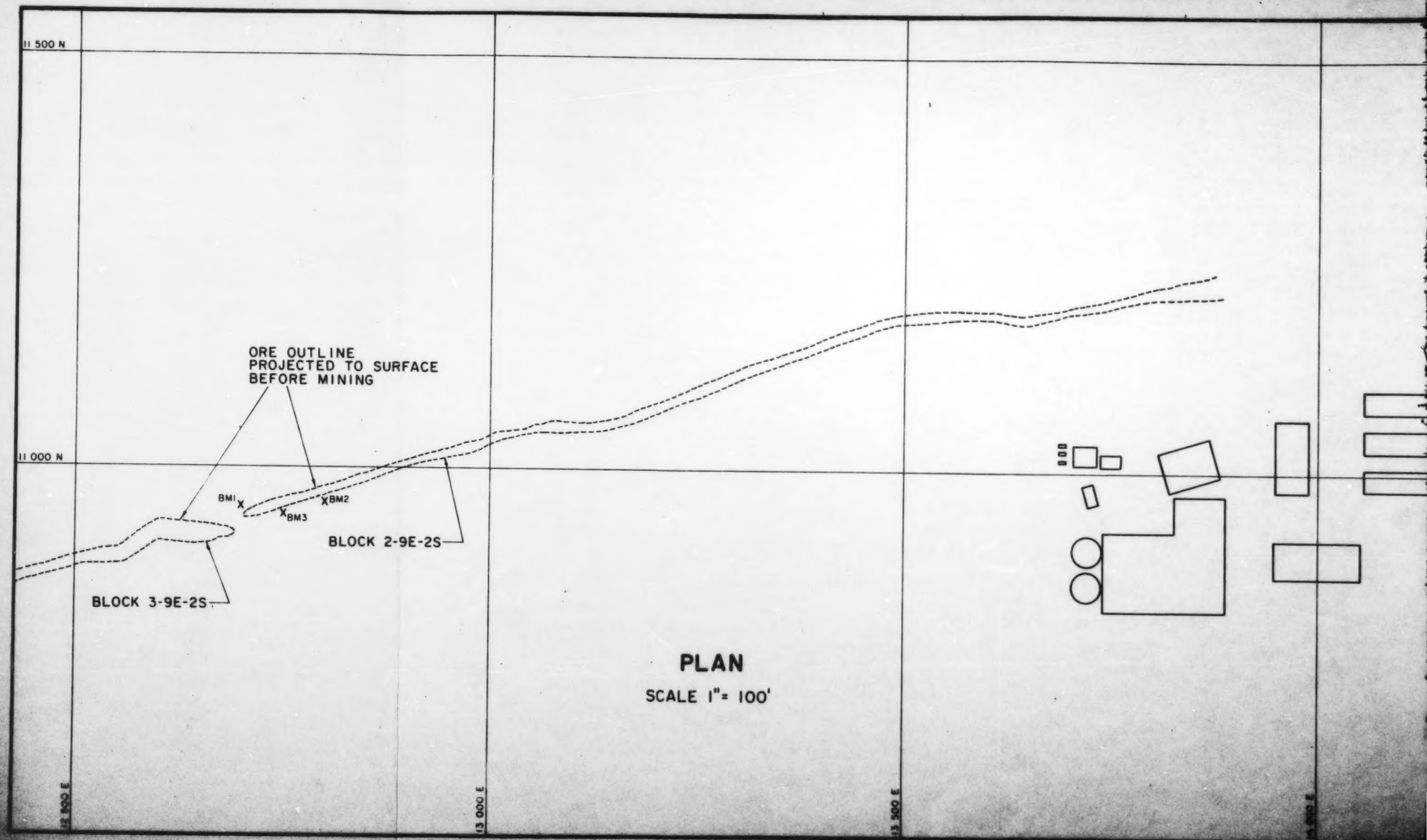


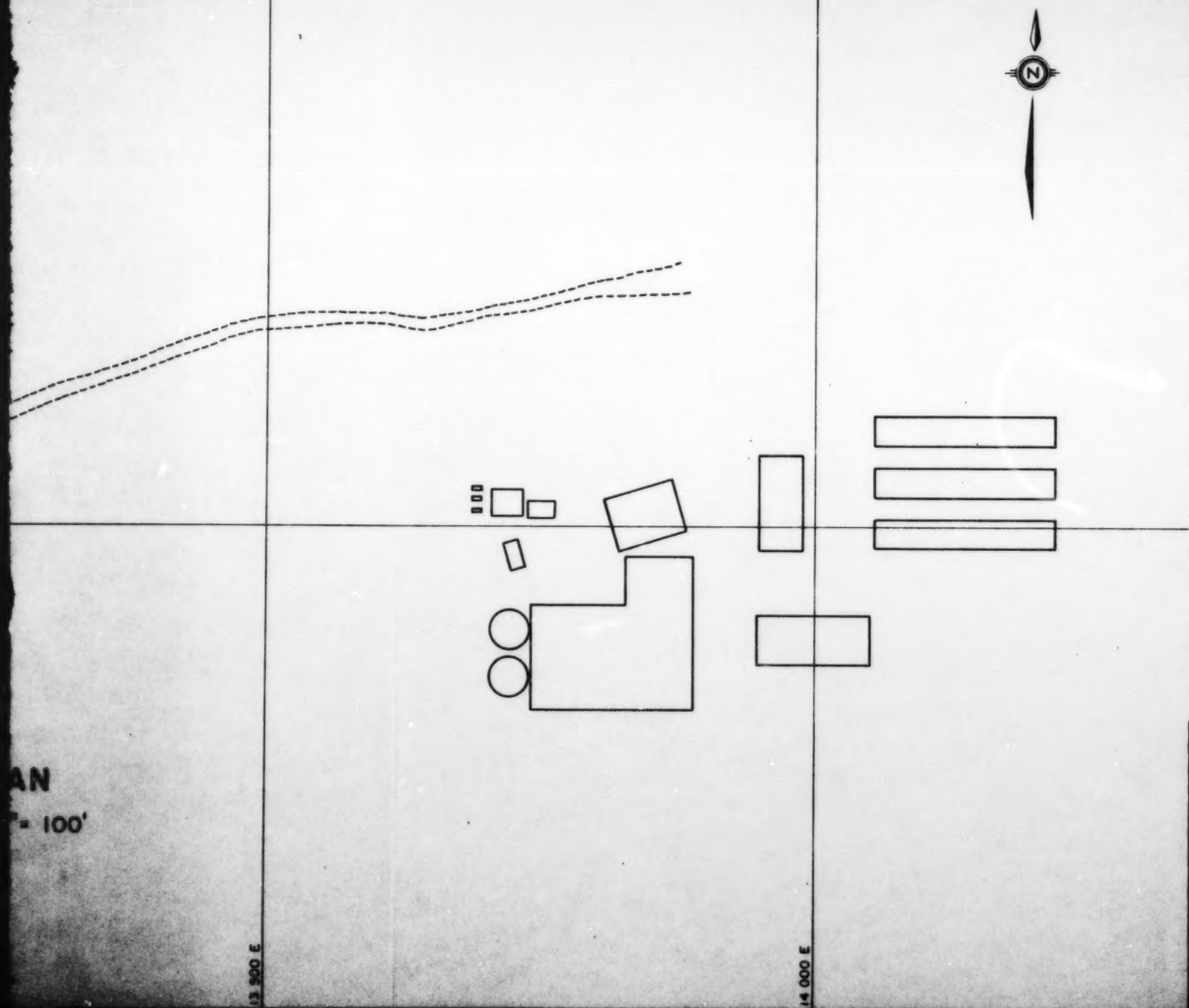
Trow

D.S.S. CROWN PILLAR INVESTIGATION
BOUSQUET MINE
MALARTIC, QUEBEC.

PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 8

SEPTEMBER 1989


11 500 N
11 000 N
12 500 E
13 000 E
13 500 E
14 000 E
ORE OUTLINE
PROJECTED TO SURFACE
BEFORE MINING
BM1
BM2
BM3
BLOCK 2-9E-2S
BLOCK 3-9E-2S
PLAN
SCALE 1"= 100'



AN
= 100'

LEGEND

X PROPOSED BOREHOLE LOCATION

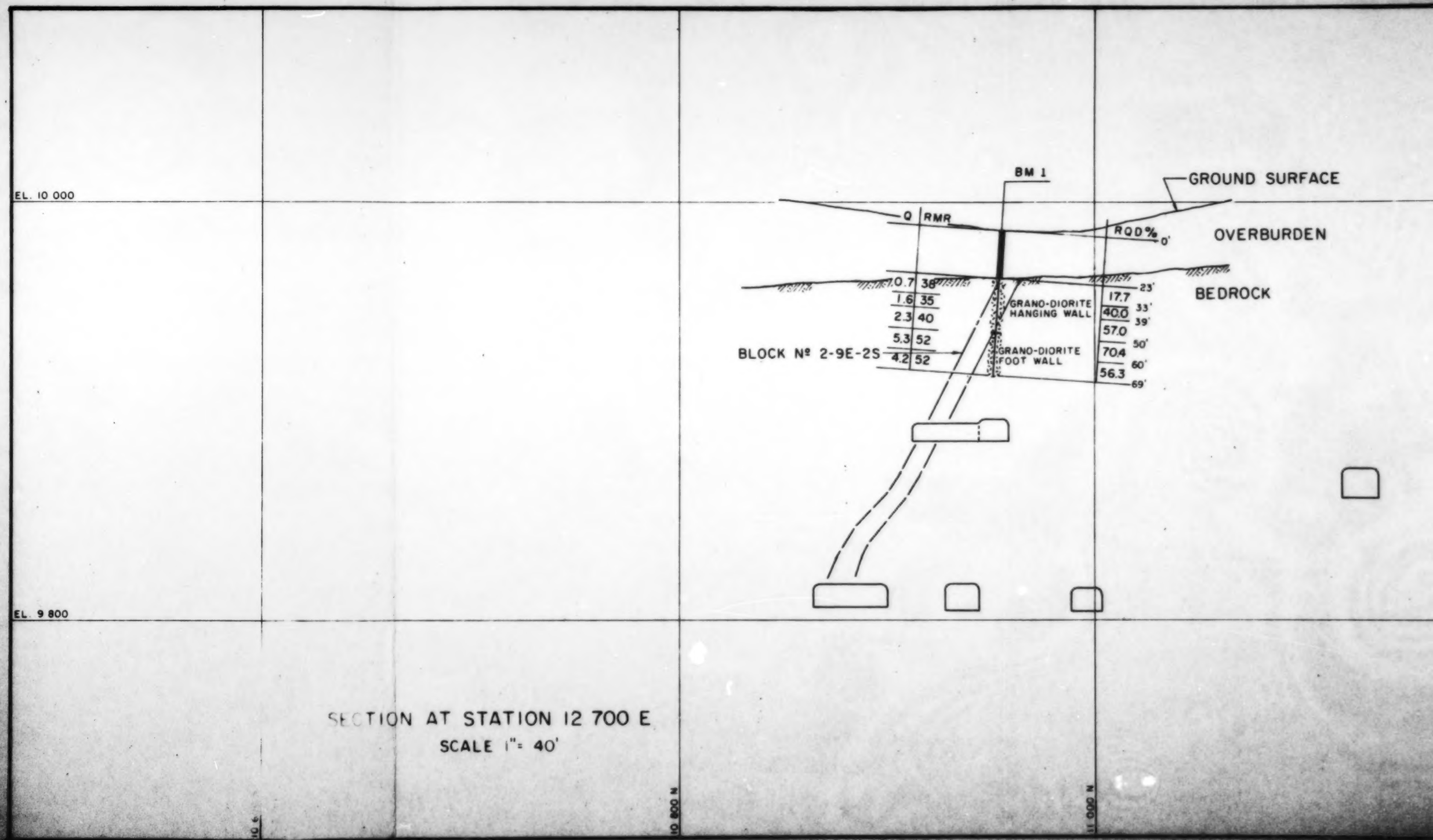
NOTES: 1) Plan traced from drawing supplied by Les Mines Belmoral Ltée.

Trow

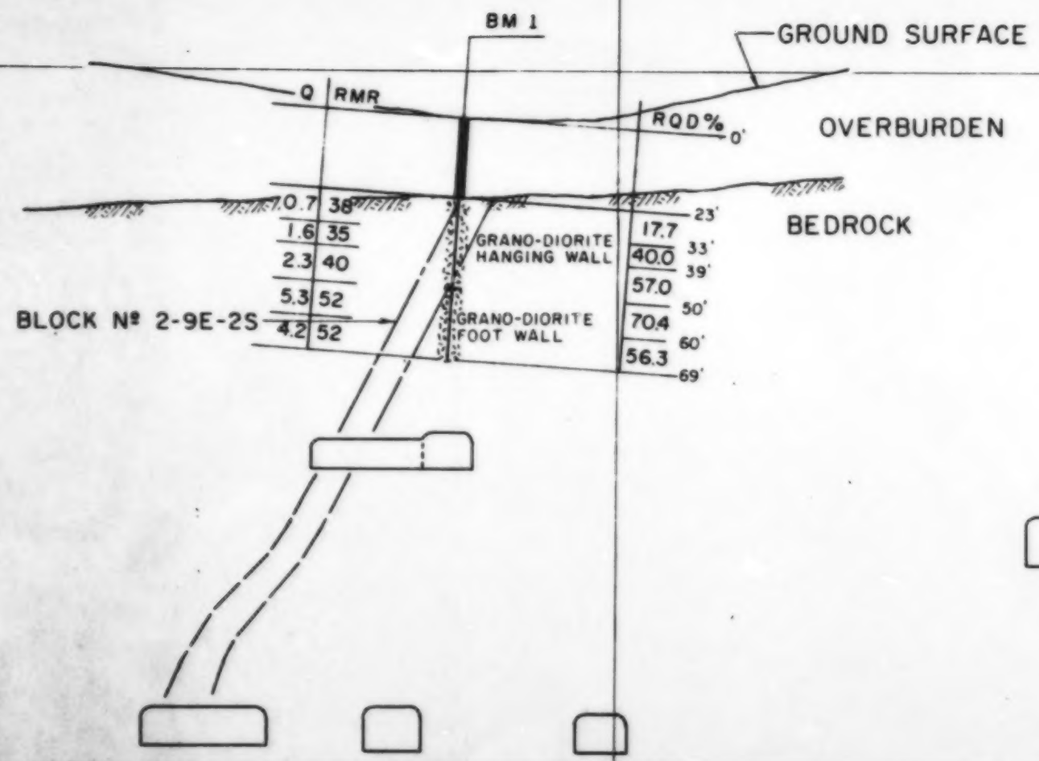
D.S.S. CROWN PILLAR INVESTIGATION
FERDERBER MINE
VAL D'OR, QUEBEC.

PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 9

SEPTEMBER 1989



SECTION AT STATION 12 700 E
SCALE 1" = 40'

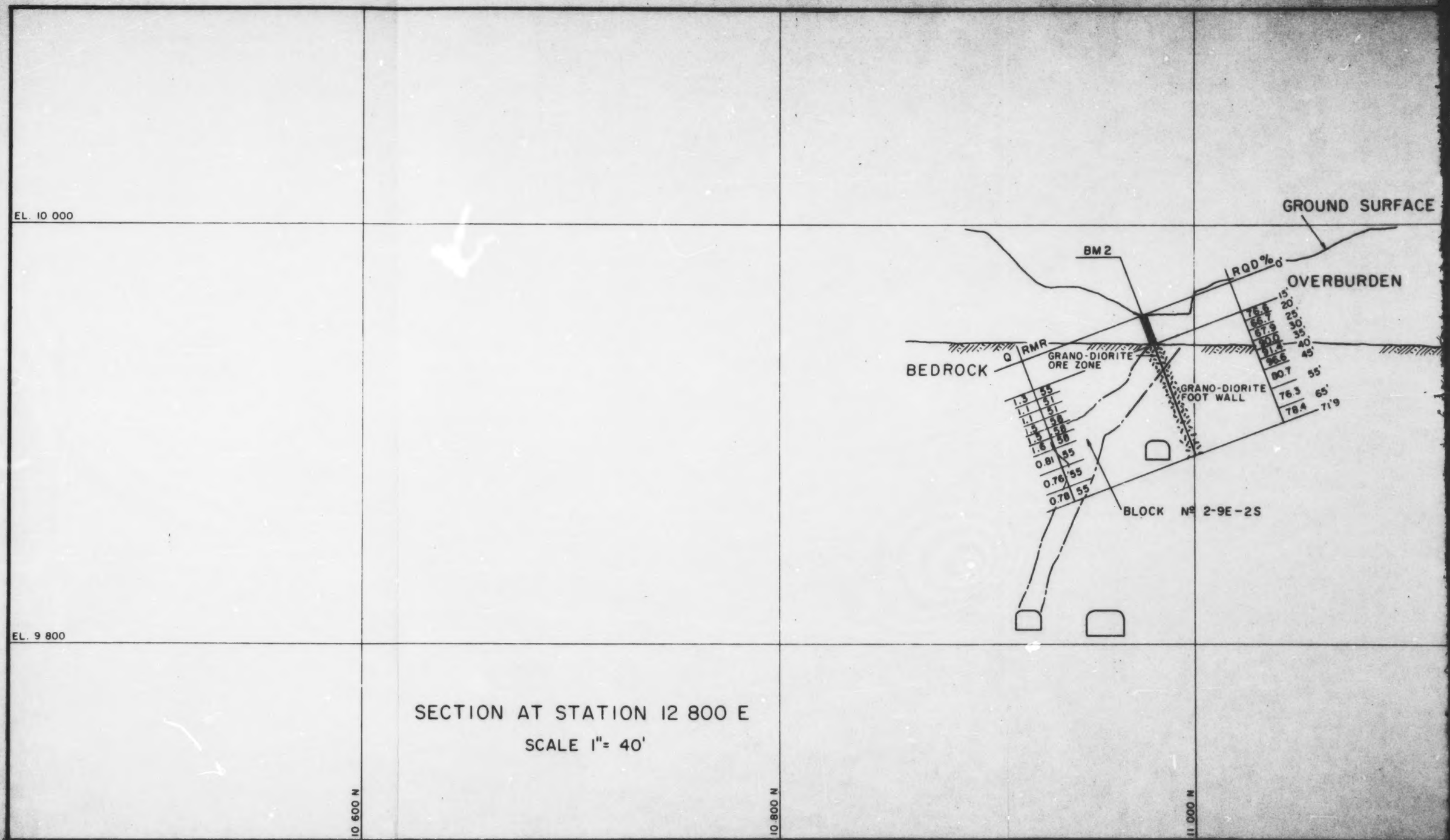


Trow

D.S.S. CROWN PILLAR INVESTIGATION
FERDERBER MINE
VAL D'OR, QUEBEC.

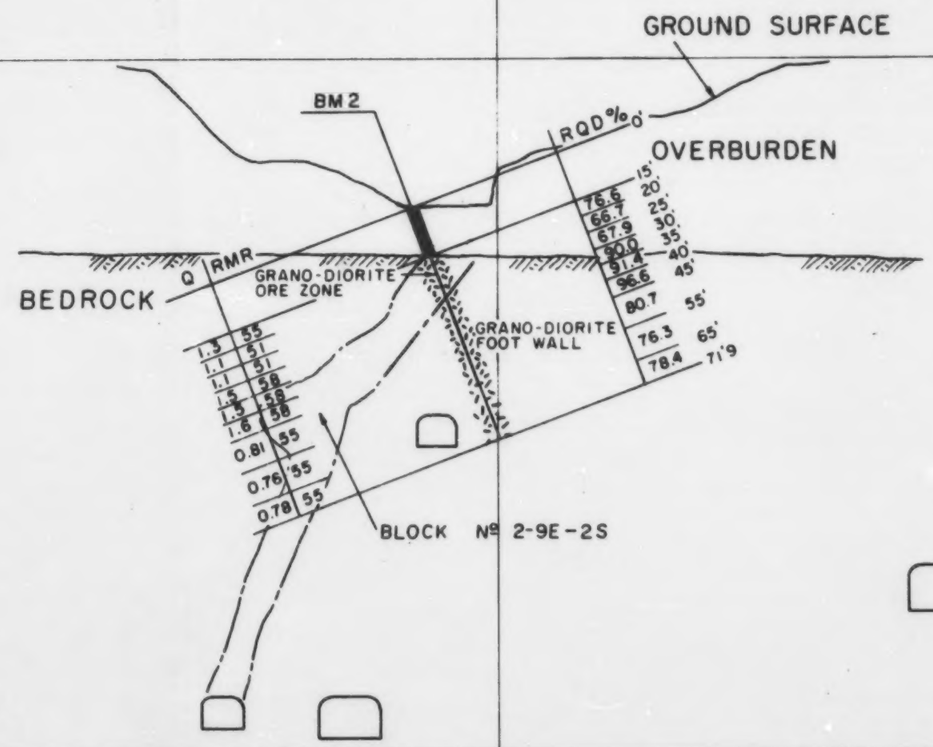
PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 10

SEPTEMBER 1989



SECTION AT STATION 12 800 E

SCALE 1"= 40'



O E

10 800 N

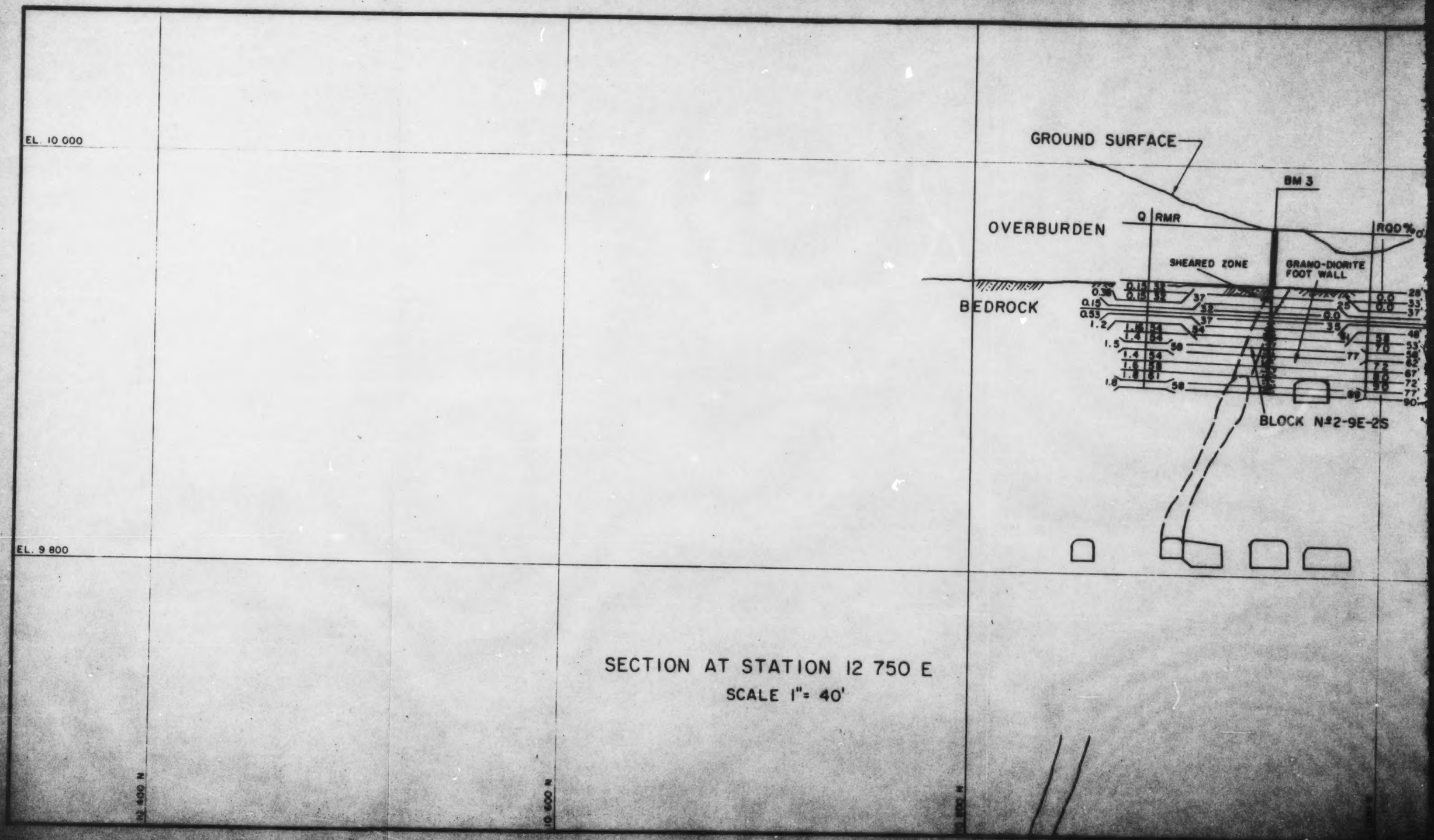
11 000 N

Trow

D.S.S. CROWN PILLAR INVESTIGATION
FERDERBER MINE
VAL D'OR, QUEBEC.

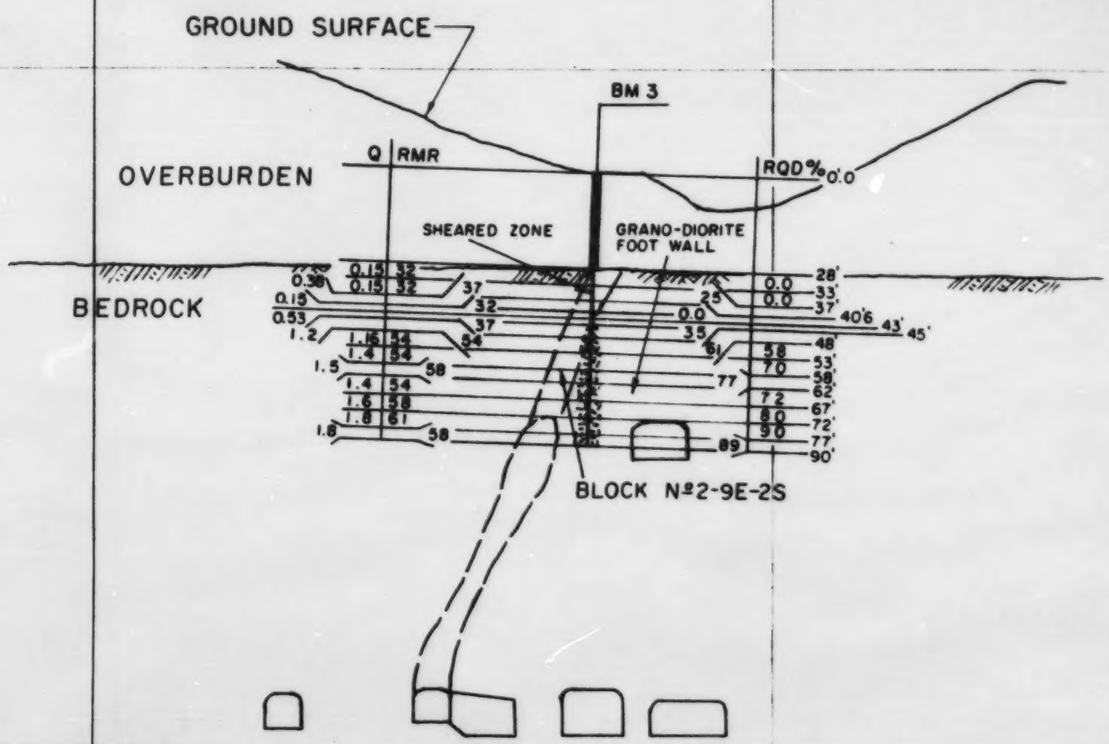
PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 11

SEPTEMBER 1989



SECTION AT STATION 12 750 E

SCALE 1"= 40'



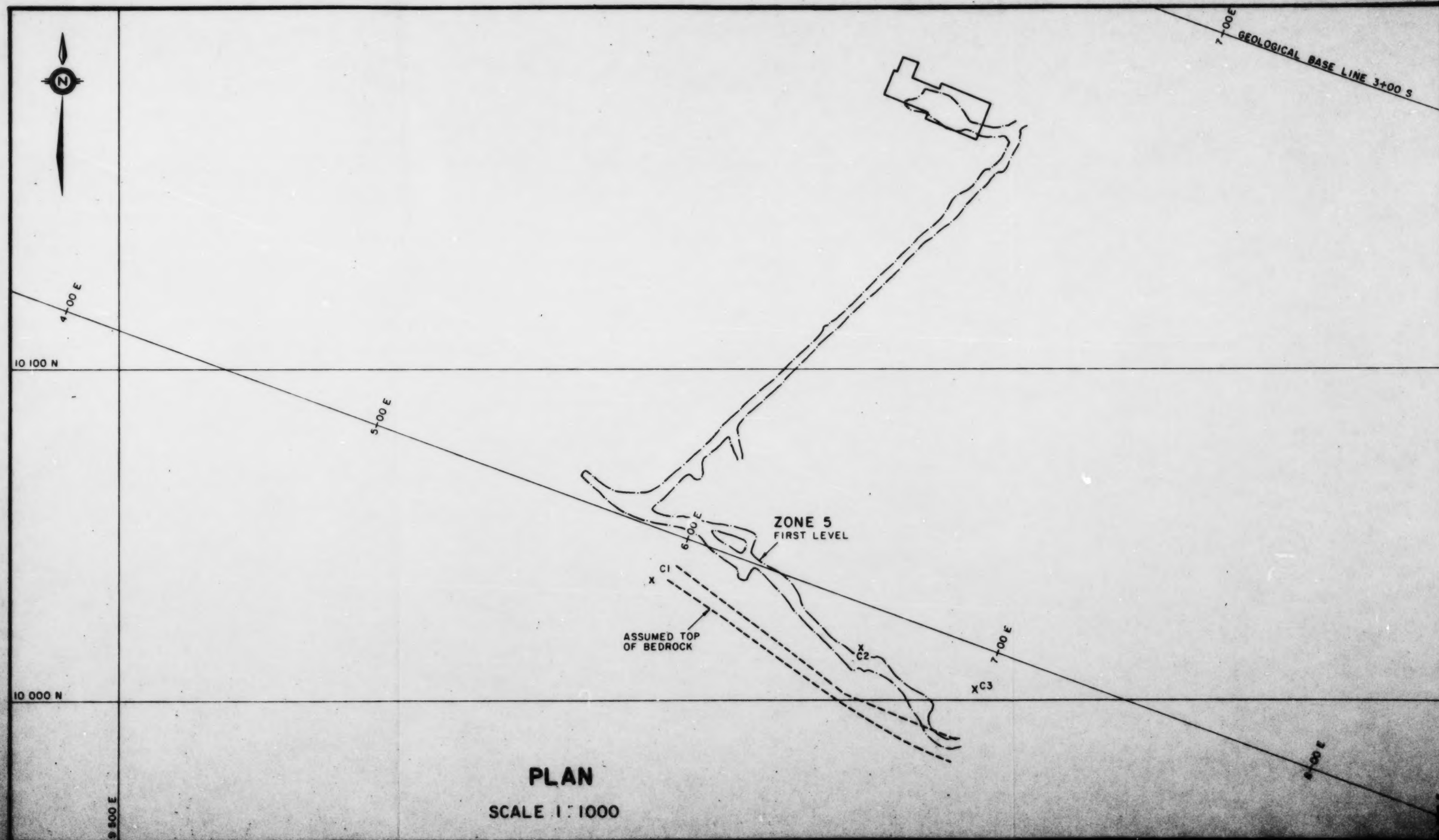
AT STATION 12 750 E
SCALE 1"= 40'

Trow

D.S.S. CROWN PILLAR INVESTIGATION
FERDERBER MINE
VAL D'OR, QUEBEC.

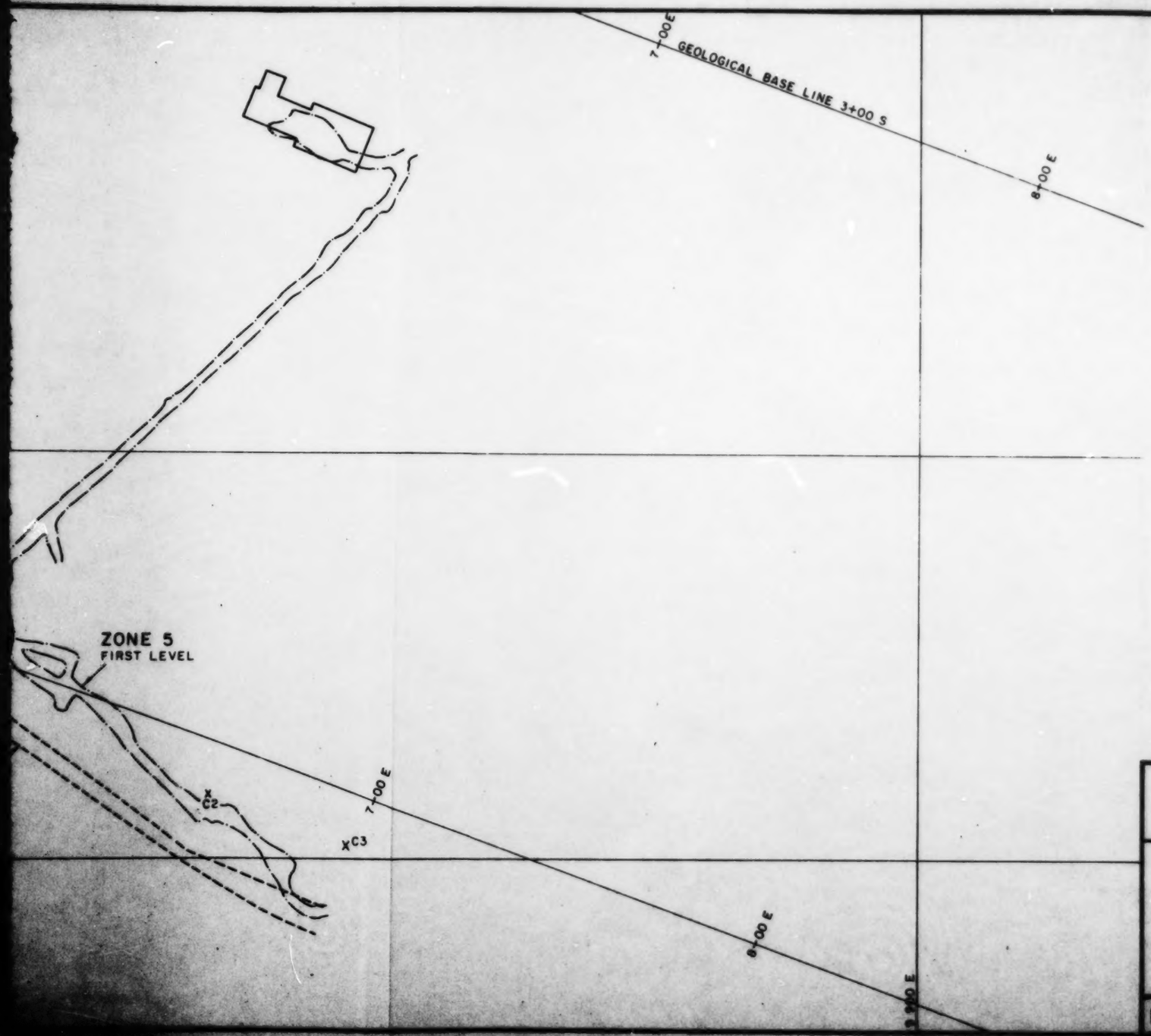
PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 12

SEPTEMBER 1989



PLAN

SCALE 1" 1000



LEGEND

X PROPOSED TEST HOLE LOCATION

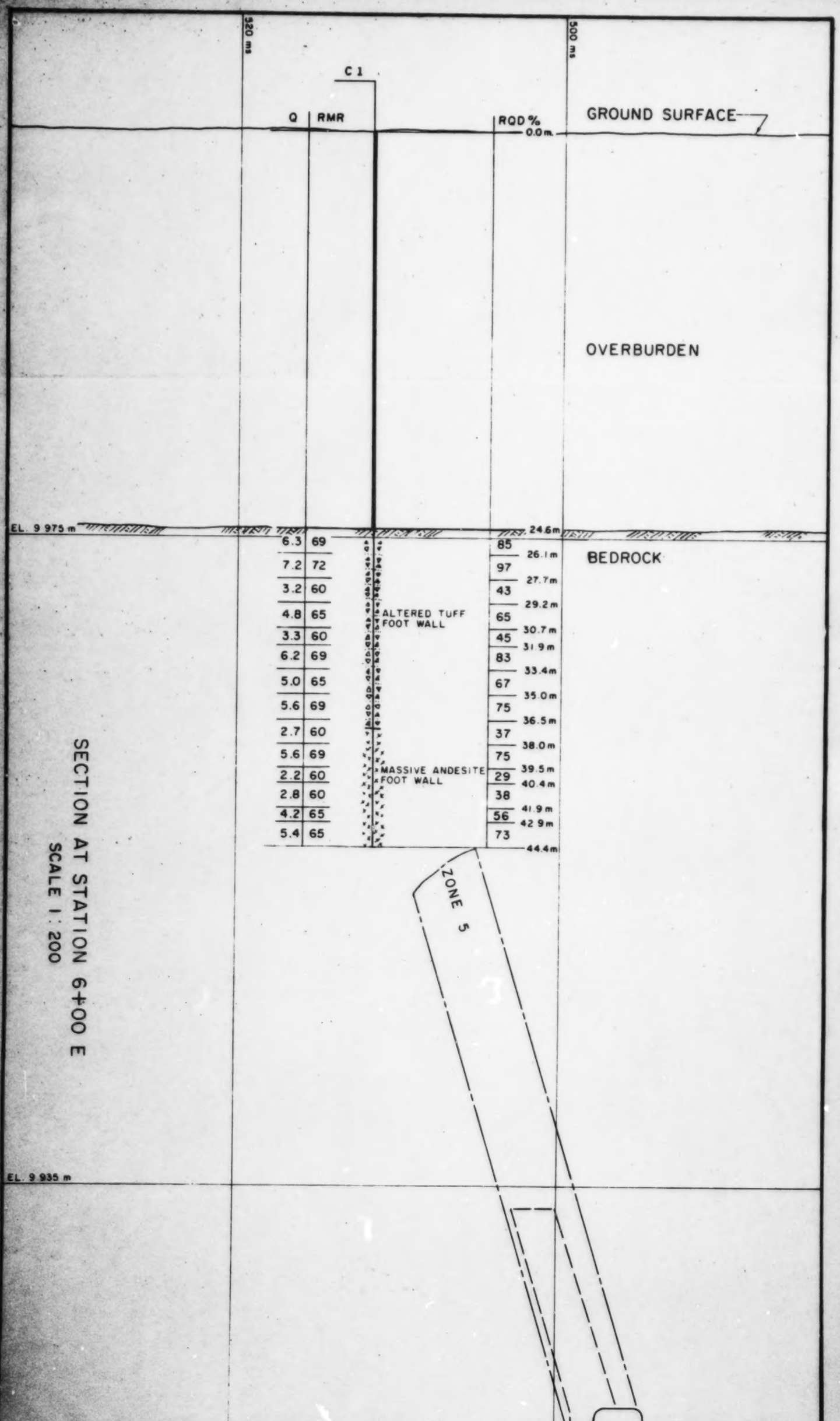
NOTES: 1) Plan traced from drawing no. 02-514-272 supplied by La Societe Miniere Louvem Inc.

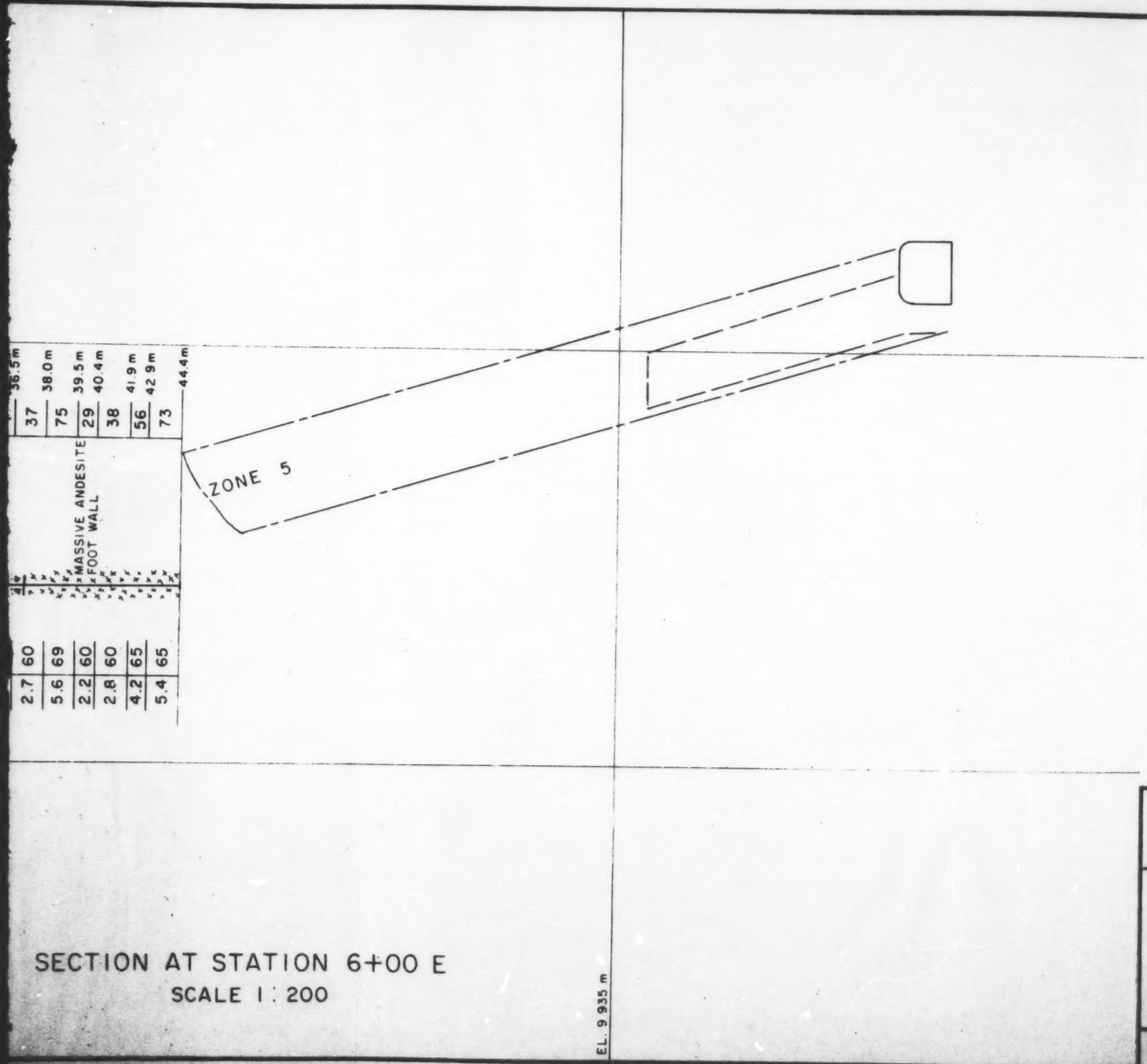
Trow

D.S.S. CROWN PILLAR INVESTIGATION
CHIMO MINE
LOUVICOURT, QUEBEC.

PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 13

SEPTEMBER 1989


320 mS
500 mS
C 1
Q
RMR
RQD %
0.0 m.
GROUND SURFACE
OVERBURDEN
EL. 9 975 m
24.6m
BEDROCK
6.3 69
7.2 72
3.2 60
4.8 65
3.3 60
6.2 69
5.0 65
5.6 69
2.7 60
5.6 69
2.2 60
2.8 60
4.2 65
5.4 65
ALTERED TUFF
FOOT WALL
MASSIVE ANDESITE
FOOT WALL
85
26.1m
97
27.7m
43
29.2m
65
30.7m
45
31.9m
83
33.4m
67
35.0m
75
36.5m
37
38.0m
75
39.5m
29
40.4m
38
41.9m
56
42.9m
73
44.4m
ZONE 5
EL. 9 935 m
SECTION AT STATION 6+00 E
SCALE 1 : 200



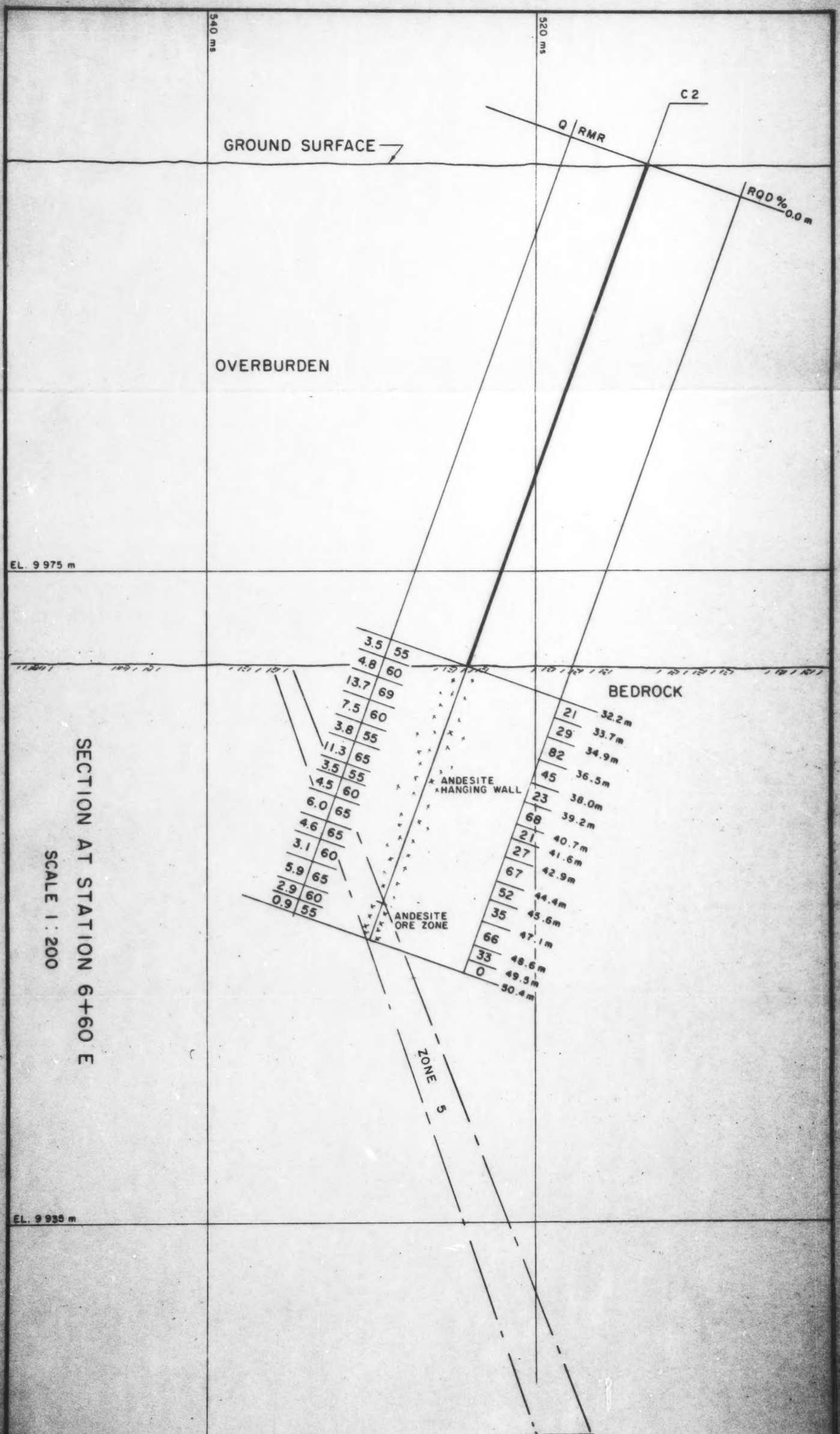
SECTION AT STATION 6+00 E
SCALE 1 : 200

Trow

D.S.S. CROWN PILLAR INVESTIGATION
CHIMO MINE
LOUVICOURT, QUEBEC.

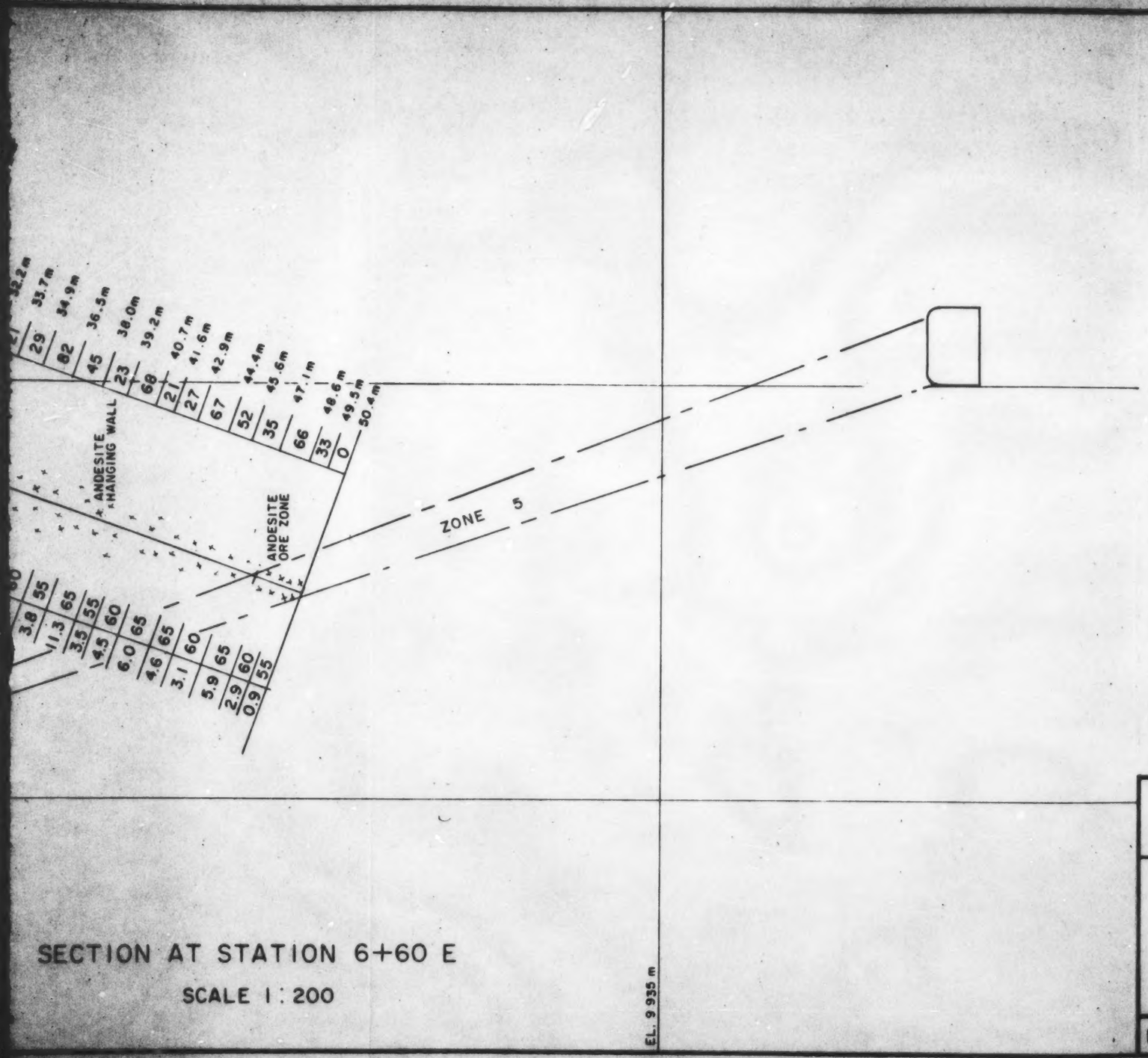
PROJ. S02095R	DATE NOVEMBER 1989	DWG. No. 14

SEPTEMBER 1989



SECTION AT STATION 6+60 E

SCALE 1 : 200



SECTION AT STATION 6+60 E

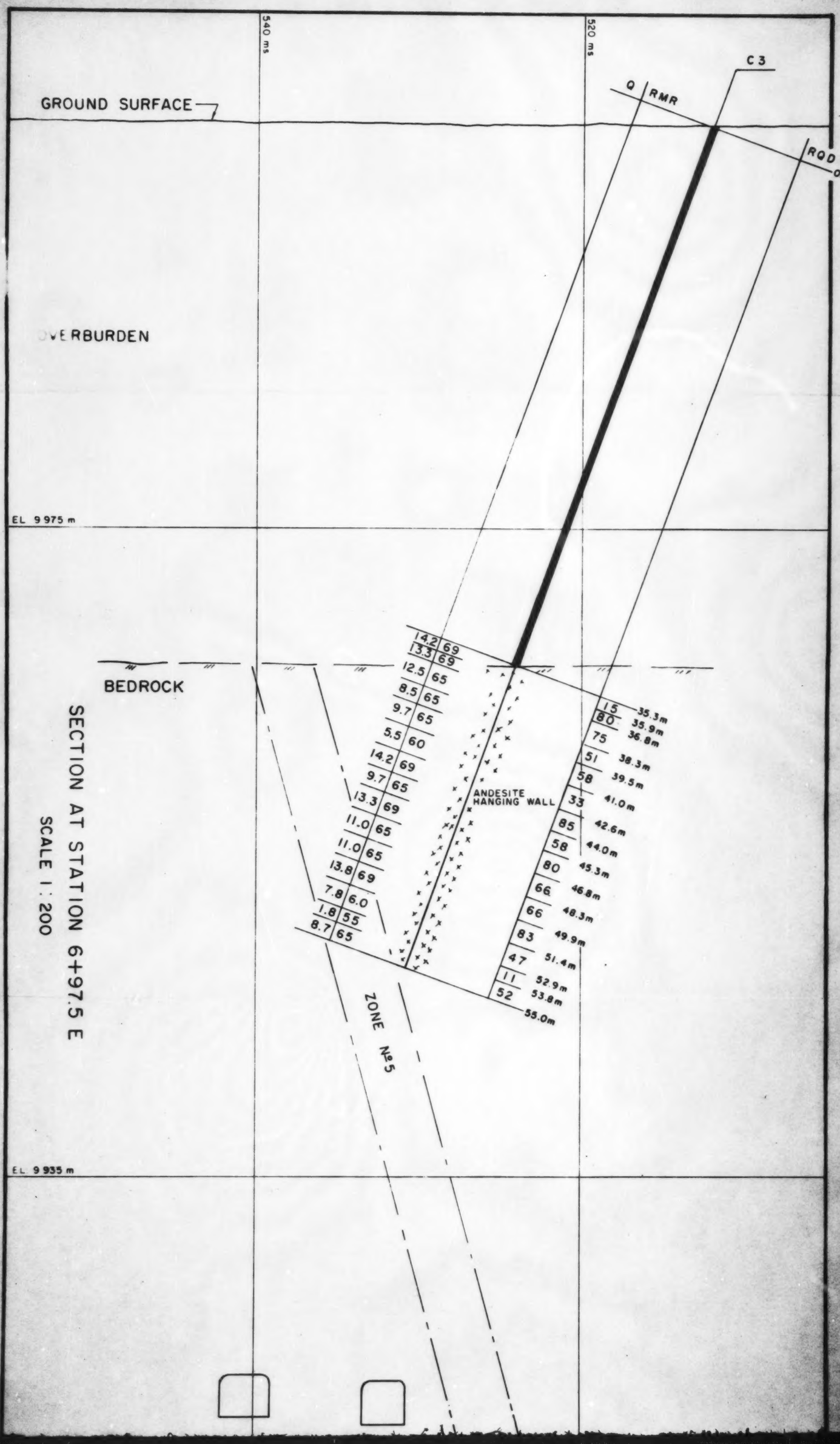
SCALE 1 : 200

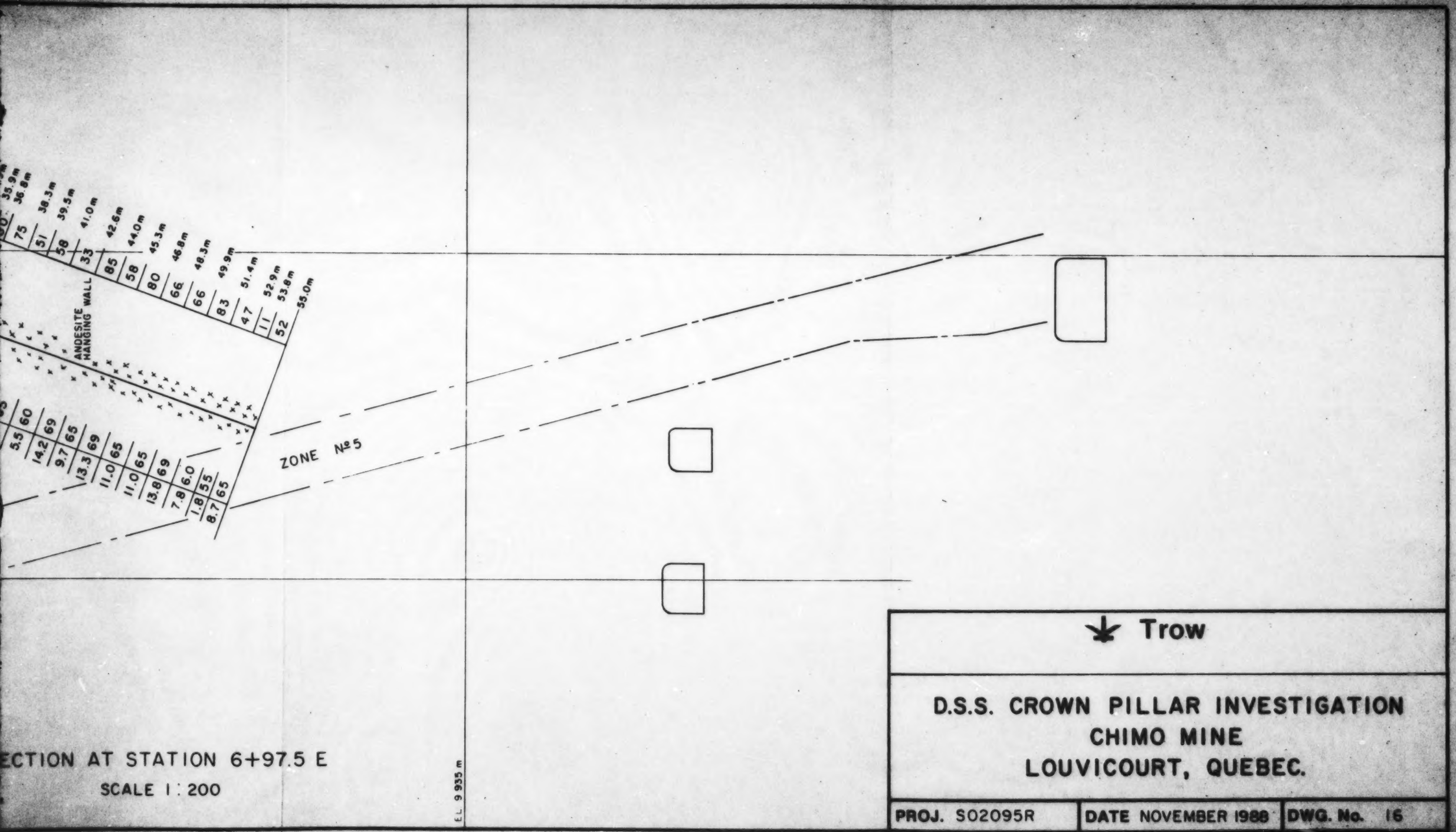
Trow

D.S.S. CROWN PILLAR INVESTIGATION
CHIMO MINE
LOUVICOURT, QUEBEC.

PROJ. S02095R	DATE NOVEMBER 1988	DWG. No. 15

SEPTEMBER 1989


540 ms
520 ms
C3
Q
RMR
RQD %
00m
GROUND SURFACE
OVERBURDEN
EL 9 975 m
BEDROCK
14.2 69
13.3 69
12.5 65
8.5 65
9.7 65
5.5 60
14.2 69
9.7 65
13.3 69
11.0 65
11.0 65
13.8 69
7.8 6.0
1.8 55
8.7 65
ANDESITE HANGING WALL
15 35.3m
80 35.9m
36.8m
75
51 38.3m
58 39.5m
33 41.0m
85 42.6m
58 44.0m
80 45.3m
66 46.8m
66 48.3m
83 49.9m
47 51.4m
11 52.9m
52 53.8m
55.0m
ZONE Nº5
EL 9 935 m
SECTION AT STATION 6+97.5 E
SCALE 1 : 200


35.9m
36.8m
38.3m
39.5m
41.0m
42.6m
44.0m
45.3m
46.8m
48.3m
49.9m
51.4m
52.9m
53.8m
55.0m
75
51
58
33
85
58
80
66
66
83
47
11
52
ANDESITE HANGING WALL
5.5 60
14.2 69
9.7 65
13.3 69
11.0 65
11.0 65
13.8 69
7.8 6.0
1.8 55
8.7 65
ZONE Nº5
ECTION AT STATION 6+97.5 E
SCALE 1 : 200
EL. 9.935 m
Trow
D.S.S. CROWN PILLAR INVESTIGATION
CHIMO MINE
LOUVICOURT, QUEBEC.
PROJ. S02095R
DATE NOVEMBER 1988
DWG. No. 16
SEPTEMBER 1989